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FOREWORD

This, the 184th volume of "Transactions" published by the American Institute of Mining and Metallurgical Engineers, is not only the first of the series in the new enlarged page size, but represents also a change in the subject matter of the individual volumes. Heretofore, technical papers on subjects in a distinct professional field, such as Mining Geology, Geophysics, Mining Methods, Milling Methods, Coal, and Industrial (Nonmetallic) Minerals, have been issued in separate volumes, or in some instances two or more such subjects have been covered in one volume. Because of the fact that enough papers usually did not accumulate on one of these subjects in any one year, their appearance in bound form in a "Transactions" volume was often delayed for as much as two or three years.

Beginning with the papers published in 1949, the present practice is to publish in bound form early in the succeeding year all the papers on these various subjects that are included in the field of the Mining Branch of the Institute. A similar procedure has been adopted for the Metals and Petroleum Branches respectively. Thus much more prompt publication in the "Transactions" will be secured, and further, metal mining men will have an opportunity to review, in convenient form, what the coal mining men are doing, and the accomplishments of those who concentrate or clean metalliferous ores, nonmetallic minerals, and coal will likewise be included within one cover. It is believed that the diverse groups can learn much from the practice of others who are doing similar work but in different commodity fields.

Another logical consequent change should be noted. Heretofore the date on the "Transactions" volume has been that of publication, or of the year for which publication was authorized. The date of this and succeeding volumes will be that of the original publication of the papers in the respective monthly journals. All of the contents of this present volume, for instance, were published in "Mining Engineering" during 1949, and no technical papers (which appeared as Section 3 of that journal) have been omitted. In fact, the present volume is merely an overrun printing of the technical papers section of the twelve issues of "Mining Engineering" for 1949, including all the printed discussion. This latter does not necessarily immediately follow the paper to which it applies, but reference to the index will make its position in the volume clear.

Three professional Divisions of the Mining Branch operated throughout 1949 and one, the Mining, Geology, and Geophysics Division, was organized towards the end of the year. They are responsible for the material in this volume:

Division	Chairman	Chairman Papers and Programs Committee	Chairman Auxiliary Publications Committee
Minerals Beneficiation	S. J. Swainson	Grover J. Holt	M. D. Hassialis
Coal	E. R. Price	Elmer R. Kaiser	H. P. Greenwald
Industrial Minerals	H. A. Meyerhoff	R. M. Foose	H. D. Keiser
Mining, Geology, and Geophysics	P. J. Shenon	John J. Collins	Darwin Pope*

*R. A. Pallanch acted as chairman before the Division was formed. F. A. Wardlaw had previously been chairman of the superseded Mining Methods Committee; P. J. Shenon of the Mining Geology Committee; J. B. Macelwane of the Geophysics Committee; and E. R. Borcherdt of the Health and Safety in Mines Committee.

Of the AIME Technical Publications Committee, which finally accepts all technical papers, E. M. Wise was chairman until October 15, 1949, and thereafter, E. C. Meagher; with E. J. Kennedy, Jr., secretary.

Attention is called to the fact that the publications of the Extractive Metallurgy Division, many of which are of some interest to the members of the Minerals Beneficiation Division, appear in Vol. 185 of the "Transactions", which includes all papers of the Metals Branch.

NEW YORK, N. Y.
FEBRUARY 3, 1950

EDWARD H. ROBIE
Secretary

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Underground Anemometry*

BY CLOYD M. SMITH,† Member AIME

A FEW years ago, the Ventilation Committee established the practice of presenting one topic each year for discussion at the annual meeting. The practice has met good response on the part of committee members and I suggest that it be continued. The topic chosen for this year, "Underground Anemometry," is a topic which has bothered me for more than 20 years. It seems to me that the coal industry is content to rely on slipshod methods for measuring the rate of flow of air underground, so I prefaced my discussional charge to committee members with the statement that I regard air measurements made in the usual way, with hand held anemometer, as no good. Agreements and disagreements came in from more than a dozen engineers, some of whom are with operating companies, coal and metal; some with manufacturers; others with government agencies.

The statement was accompanied by a questionnaire on the use of the rotating vane anemometer and by one describing two methods of using a mechanically held anemometer. The questionnaire will be considered first. The questionnaire and statement are as shown on pages 5 and 6, the committee members and respondents are given on page 4, and the general comments of the latter on page 5.

Questionnaire

1. *Has your company or agency issued written instructions for care and use of anemometers? If so, please enclose a copy with reply.*

Only one answer, McElroy's, was affirmative. It gave reference to Bureau of Mines publications[1,2] which recommend the hand held anemometer for rough measurements and indicate that an accuracy of 5 pct can be had if calibration and method factors are used.

Mathews said that instructions are principally oral while Maize reported that state inspectors of his department are well trained in use and care of anemometers.

2. *Are your anemometers calibrated periodically? If so, by their manufacturer? or by? Are calibration corrections applied to all observed mean velocity readings?*

Only one respondent, Lee, answered negatively as to calibration. This means that anemometers are generally calibrated but the questionnaire failed to ask how often this is done. As no one volunteered the information, we have no data on this point. In six cases the instruments are sent to their manufacturers for calibration, but Krickovic reports that his company limits manufacturers' calibrations to anemometers which are used by operating personnel; those used by the engineering department being calibrated by U. S. Bureau of Standards. The Anaconda Copper Mining Co. has its ventilation engineers calibrate its anemometers.

Most of the respondents say that a calibration correction is applied to each mean velocity reading, but Krickovic limits this to surveys made by the engineering department. Since Lee does not calibrate, he has no correction to apply. Maize reports that his department has its anemometers calibrated but does not apply corrections.

3. *Do your men hold the anemometer by hand in measuring air flow? for 1 min? or traverse the section? for 1 min? or at? points for 5 sec each?*

Of 10 replies, 6 were "yes," 3 were "no," and one was "seldom" with respect to holding by hand. Among the six hand holders, four hold in a central position in the measuring section for a minute, except that two of them, Krickovic and Matthews, traverse the section by hand for survey or fan test. Their operating personnel hold by hand, centrally, for routine measurements. McElroy sometimes traverses with hand-held anemometer in rapid survey work.

4. *If the anemometer is not held by hand, how is it supported?*

Augustadt supports the anemometer on an adjustable rod, Condon on "a rod of sufficient length to reach all points with observer standing in one position throughout traverse and at arm's length from plane of traverse." I presume that arm's length must be interpreted liberally enough to allow for arm movement, otherwise it would be impossible to manipulate the anemometer throughout the traverse section. Mancha upholds Condon in this method of traversing. Glanville hangs the anemometer on the end of a 4-ft staff by the hasp at the top of the anemometer frame. McElroy mounts it on the end of a rigid square shaft, 12 in. long, the staff being at right angles to the axis of the instrument. He traverses the section in two halves, holding the anemometer 3 feet from his body.

New York Meeting, February 1948.
TP 2502 F. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before April 1, 1948.

* Statement of personal views presented at the annual meeting pursuant to study of replies to questionnaire submitted to members of the Ventilation Committee of the Coal Division, and to others. Manuscript received May 17, 1948, revision, Sept. 8, 1948.

† Chairman, Ventilation Committee, Coal Division AIME; Mining Engineer, Washington, D. C.

[1] References are at the end of the paper.

5. *Is each initial determination repeated immediately after it is taken, as a check? If so, can you supply data on the self consistency of such measurements, i.e., how well do the repeat runs check the first runs?*

Most observers take check readings, but Matthews limits them to tests for new fan installations. As to consistency of results, Beckwith reports thousands of checks within 5 pct, Krickovic reports checks within 10 to 15 pct, pointing out that perfect checks would not ensure accuracy since both readings might be equally wrong. Matthews reports checks on 4 or 5 readings within 5 to 10 pct with the anemometer held "on a stick." The best consistency is reported by Condon as follows:

Mean velocity range, fpm	under 500	500–1500	over 1500
Consistency within, pct	0.5	1.0	2.0

Field notes submitted by Glanville covering 11 paired traverses of as many sections at mean velocities ranging from 264 to 1250 fpm show 5 pairs in which the readings are within 1 pct of each other; 3 pairs within 2 pct; and 1 pair each within 3, 7 and 9 pct. The last two were known to be affected by haulage movements. Richardson reports that variations are usually less than 5 pct, commonly 2 pct, with a stick-mounted instrument. Under similar conditions McElroy reports checks within 1 pct for good conditions and within 2 pct for poor conditions by an experienced operator.

6. *Has your company or agency issued instructions for choice of measuring section, covering such points as desirable approach and departure conditions, freedom from obstructions, preferable shapes and dimensions, and methods of determining cross-sectional area? If so, please enclose a copy with reply.*

Only Beckwith reported issuance of instructions for selection and measurement of air measuring sections and those instructions are mainly oral.

7. *Do your men take air measurements in or near regulator openings? If so, directly in the opening or how far downstream from it?*

There were four flat negatives and tbree flat positives in the answers to this question. In one of the latter cases reported by Beckwith the measurements are made in, not near, the regulator opening. Maize states that measurements are taken in the regulator opening only when a small volume of air is flowing and that, if velocity of the air is high as it enters the regulator, measurements are taken far enough away for the readings not to be affected. No indication is given as to how far from the regulator this must be, whether upstream or downstream, nor is there any indication as to what are regarded as high and low velocities and quantities. Krickovic reports taking readings near regulators at times, but ordinarily from 100 to 200 ft downstream according to the velocity. Matthews states that they are not taken within 50 ft of a regulator, while McElroy reports that regulators are avoided if possible but that door frames are acceptable. Readings are taken at the downstream edge of the regulator or frame, when these locations are used.

9. *How are sections measured to determine their areas? By tape only? by plumb bob and tape? or by special instrument or methods? If special methods are used, please describe.*

Most respondents measure sections for determination of cross-sectional area with tape only. Condon measures irregular areas by using the maximum width and average verticals taken at 1-ft intervals along the width. Glanville uses plumb bob and tape, then plots the cross section and determines its area by planimeter. McElroy uses plumb bob and tape for fan tests and a specially designed pantograph[3] for research. A comparable research method is described by Callen and Smith.[4]

10. *General remarks.*

Glanville reports close checks in traversing with anemometers and Pitot tube at high velocities and cites two instances:

(a) A Pitot tube traverse in the casing of a high-speed high-pressure propeller-type fan and an anemometer check in a carefully planimetered section 60 ft outby the fan.

Pitot tube air volume	61,100 cfm
Anemometer air volume	59,700 cfm
Difference	2.3 pct

(b) Exhaust of a fume collecting duct 14 by 14 in. square.

Pitot tube air volume	4,805 cfm
Anemometer air volume	4,897 cfm
Difference	1.1 pct

Richardson states: anemometers are calibrated at the end of a small diameter duct against velocity head by Pitot tube. The instruments are cleaned and lubricated as necessary by a watchmaker. A dirty anemometer can give very erroneous readings. High and low speed anemometers should be used according to air velocities. Continuous movement through a traverse does not check out as well as point by point traversing against Pitot tube at a measuring station.

Statement on Traversing

The questionnaires were mailed out with a statement titled "Underground Anemometry" (for the complete statement see page 6) which described two methods of traversing with mechanically supported anemometer and recommended that the two methods be tested against Pitot tube traversing. The major part of this statement is reviewed in three sections in the light of group discussion as follows:

1. The rotating vane anemometer has been the principal instrument for measuring rates of air flow in mines since the earliest days of systematic ventilation, and it has won an indispensable role in that field. However, it is one of the most abused of engineering instruments, and a shamefully small amount of thought has been devoted to its use. The principal abuse of the anemometer lies in its being held in the hand during registration, a method which brings the operator's body so close to the measuring section and to the instrument as to disturb the flow of air through the section and past the instrument and thereby to distort the results. Whether the hand-held anemometer is held in one position or traversed about the section, there is sufficient evidence to rule out this method of air measurement for any but the roughest estimates of rates of flow. Nevertheless, the anemometer is held by hand by company, state, and federal inspectors in thousands of mines daily.

These contentions with respect to manual anemometry are questioned by Beckwith who asks for the evidence which rules out this method from the standpoint of accuracy. This evidence was presented 20 years ago by Callen and Smith.[5] As far as I know, their work is the only published comparison of hand anemometry with another method, in this case Pitot tube traversing. The reports show traversing

with the hand-held anemometer to have given average mean velocities which were 19 to 29 pct greater than those determined with Pitot tube at three traverse sections at mean velocities in the range of 500 to 1100 fpm. More than 100 anemometer traverses were run, holding the anemometer by hand and traversing first one-half then the other half of the section to keep the instrument at arm's length while it was registering. Different anemometers and observers were used at sections of varied shapes and rates of flow. Not only were the anemometer results surprisingly in excess of the Pitot tube results, but they were far less consistent. As indicated in the published report, they were so bad as to discourage further anemometer work. However, Beckwith, Krickovic, and McElroy report consistent results with hand traversing, although Krickovic recognizes that self-consistency of a method has no bearing on its accuracy, in that it may be consistently wrong as well as consistently right.

2. If accurate results are to be had, the anemometer must be supported mechanically and the measuring section must be traversed. Traversing is usually done by holding the anemometer for 5 sec in each of several well-distributed positions in the measuring section. Two methods of mechanically supporting the anemometer in the section during traversing are in use. In one the instrument is hung from the end of a long stick or rod by a hasp at the top of the anemometer frame in such a way that the instrument's axis of rotation and the supporting rod are in a vertical plane parallel to the direction of flow. In traversing, the operator moves the instrument from point to point in the measuring section, while standing well downstream from the section. As nothing but the anemometer is introduced into the measuring section, this method will be referred to as the clear section method of traversing.

Krickovic insists that a section which is to be so traversed must be specially prepared, but states that many years of experience have shown that good results can be expected in any well chosen traverse section with only the preparation which would normally be given a section before it is measured, such as checking the top, smoothing the floor and knocking loose coal from the ribs. Column points are marked 2 to 4 ft apart across the roof, according to the width of the section and the anemometer is held under each roof mark for 5 sec at each point in a succession of points which are spaced approximately 2 ft apart vertically. The anemometer progresses from column to column until the traverse is completed. The method has proved to be widely applicable, expeditious, and simple.

Condon raises the point that gravity suspension permits the anemometer to swing out of plumb so that its axis is inclined to the direction of flow, and that swinging may introduce pulsating velocities through the instrument. Whether this introduces errors of appreciable magnitude awaits investigation. McElroy proposes moving the anemometer slowly up and down along vertical lines 1 or 2 ft apart as an alternative to holding it in successive points in each column. Whether this is preferable to point traversing should also be determined.

3. In the other method[6] the instrument is rigidly attached to the end of a long stick or rod by a screw socket on the cylindrical frame of the anemometer in such a way that the rod is perpendicular to the axis of vane rotation. In traversing, the operator stands centrally in the passage close enough to the measuring section to hold the rod and instrument in the section and to move the anemometer from point to point without shifting his feet and with minimum bodily movement. Inasmuch as he must essentially stand in the section, facing the air stream, this will be referred to us the obstructed section method.

Krickovic feels that equally good results could be obtained by extending the arm instead of using a rod. However, in the view of the author, the operator would have to move his body to cover any but unusually narrow sections whereas the crux of the method is that the operator should remain motionless so that a fixed rate of flow will prevail in each point of the section. The more nearly he stands still the more nearly the flow pattern remains constant throughout the traverse. While the pattern differs from the normal that exists with no one in the passage, the difference is immaterial so long as the pattern remains fixed during traversing and so long as the section is adequately traversed to give a true mean velocity. Necessary movement of the hands and arms in traversing and bodily movements are minimized.

Condon points out a source of error in inability of the operator to gauge visually the angle between the axis of the instrument and the direction of flow and to maintain parallelism. This

Table 1 . . . Classification of Methods of Traversing with Mechanically Held Anemometer

Basis	Reference	Essence	Name
Approach of operator to section and degree of consequent bodily interference with flow.	I	Operator 4 to 5 ft downstream from section; minimum feasible interference with flow through section but changing pattern.	Clear section.
	II	Operator 1 to 2 ft downstream from section; interference with flow through part of section but fixed pattern.	Obstructed section.
	III	Operator in section in 2 positions, successively; interference with flow through part of section; 2 patterns; traversing in clear part of section only.	Obstructed section.
Method of anemometer support.	I	Gravity suspension of anemometer by hasp. Supporting rod (ca 6 ft) and axis of instrument are in vertical plane which is perpendicular to traverse section.	Suspended anemometer.
	II	Rigid attachment of anemometer frame to end of (ca 6 ft) rod. Rod in traverse section. Axis of instrument perpendicular to rod.	Attached anemometer.
	III	The same except short rod (ca 1 ft).	Attached anemometer.
Positioning of operator.	I	Operator 4 to 5 ft downstream from section immediately behind each traverse column, ordinarily 4 or 5 operator positions.	Multiple position.
	II	Operator 1 to 2 ft downstream from section, midway of entry. Negligible bodily movement.	Single position.
	III	Operator in section first to 1 side of instrument then to other; crouches to shift between positions.	Dual position.

error is akin to the error he cited in the other method due to swing of the anemometer. Both await valuation.

McElroy (Table 1 reference No. III) proposes a third method of traversing with mechanically supported anemometer. That is to screw the instrument onto a short rod, instead of a long one as in the second method, and to have the operator stand in the section and hold the anemometer at arm's length while slowly backing across the section to about midsection, then to change position by crouching and passing the anemometer above his body, thereafter to advance slowly, moving the anemometer up and down to complete the traverse.

Mancha takes exception to the nomenclature used in designating the methods on the grounds that the term "obstructed section method" is prejudicial. Two other bases of nomenclature suggest themselves:

1. The method of support of the anemometer.

2. The number of positions occupied by the operator during traversing.

Two methods of mechanical support are under consideration, suspension and attachment, while three classes of operator positioning are involved. The situation is analyzed in Table 1. Since the positional basis of classification is the only one that gives three names, single, dual and multiple-position, for the three methods, its use is suggested pending the devise of a better system.

Proposal for Investigation

As to choice of traversing methods the answer would seem to lie in direct comparison of these methods and of other methods which may be proposed, including manual holding, with results of Pitot tube traversing in a specially prepared and instrumented mine passage or wind tunnel. This is a small research project which the industry should undertake as a step toward putting its measurement of air flow on a sound basis. That it is on no such basis now is highlighted by Harrington who says:

Anemometer air measurements of the usual kind, by holding the instrument in the hand usually with the body in the air stream, are mere approximations and this is true to even a greater extent of the usual tape measuring of the cross sectional area. These two *guesses* are multiplied together and the quantity thus secured is at best a not particularly close approximation as to the actual quantity.

He goes on to say that this is not a matter of much practical concern, but I do not hold with the view that it is immaterial whether quantity measurements are 20 to 30 pct off, nor do I hold with those who insist that coal mine supervisory employees are too dumb to learn a simple accurate method of traversing if one is developed. It is my notion that any man who can properly care for and use a flame testing lamp can also do good work with a vane anemometer, once he is taught how. The investigation I propose should result in a method of underground traversing with the rotating vane anemometer which will be generally applicable to and acceptable for fan testing and ventilation surveying, and it should lead to a simple but reasonably accurate method of traversing for use by operating personnel in making routine quantity measurements as required by law; a method which will give better and more dependable results than the present haphazard work.

There is not space enough to permit further discussion nor complete treatment of the general comments which many members of the group submitted. In closing, I want to explain that I took the liberty of including some nonmembers of the committee, indeed of the Institute, in the group in view of their known interest and competence in the subject and I want to thank members and nonmembers alike for their splendid help. I also want to warn that this statement is the report of the chairman, not of the committee, and it is therefore primarily expressive of his views. No committee member has seen it nor has responsibility for it, but it is hoped that it will meet approval of the committee as a whole. It is respectfully submitted for committee disposal.

List of Committee Members and Respondents

Ventilation Committee Members:

Alfred T. Beckwith, Special Engineer, Lehigh Navigation Coal Co., Lansford, Pa.

John W. Buch, Supervising Engineer, Anthracite Mechanical Mining Section, U. S. Bureau of Mines, Schuylkill Haven, Pa.

Stephen Krickovic, Chief Engineer, Koppers Coal Co., Pittsburgh, Pa.

Raymond Mancha, Vice President, Joy Manufacturing Co., Pittsburgh, Pa.

Charles H. Matthews, Electrical and Mechanical Engineer, Susquehanna Collieries Co., Nanticoke, Pa.

F. C. Sturges, Vice President, Pennsylvania Drilling Co., Pittsburgh, Pa.

Cloyd M. Smith, Consulting Engineer, Washington, D. C., Committee Chairman.

The following who are not members of the committee contributed to the discussion on invitation:

G. L. Augustadt, Ventilation Engineer, Magma Copper Co., Superior, Ariz.

R. E. Condon, Manager of Sales, Ventilation Division, Jeffrey Manufacturing Co., Columbus, Ohio.

Leo Glanville, Ventilation Engineer, Climax Molybdenum Co., Climax, Colo.

Daniel A. Harrington, Chief, Health and Safety Branch, U. S. Bureau of Mines, Washington, D. C.

Carl Lee, Chief Engineer, Peabody Coal Co., Chicago, Ill.

Richard Maize, Secretary of Mines, Harrisburg, Pa.

* G. E. McElroy, Supervising Engineer, Mine Ventilation Section, Health Branch, U. S. Bureau of Mines, Pittsburgh, Pa.

Allan S. Richardson, Ventilation Engineer, Anaconda Copper Mining Co., Butte, Mont.

* Became member of committee March 3, 1948.

General Comments by Respondents

A. T. Beckwith—I cannot agree that holding the anemometer in your hand is no good, inasmuch as my own experience with this method has been contrarywise. Following are my viewpoints on the matter:

The standard rotating vane anemometer is a precision instrument; the application of this instrument to its function of measuring air flow is a science. Mainly because of a lack of knowledge as to what constitutes good practice, the majority of men who take anemometer readings obtain inferior results.

In recent years I have made several thousand air quantity measurements by hand-holding anemometers, and have found, by repeat measurements and the split flow method of checking results, that such measurements can be expected to consistently balance out within 5 pct. This degree of accuracy, I believe, is sufficient for all requirements in the average mine, with the possible exception of the measurements made to determine fan performance.

A first consideration toward obtaining a measurement of air flow is selection of the measuring station in the available portion of the air course. Frequently a measurement is desired in a section of the workings wherein but a short length of air course is available because of the proximity of one split to another. When, of necessity, the station has an irregular outline, has nearby obstructions, or is in a curved air course, the undesirable conditions must be recognized as such. Measurements of the splits at satisfactory locations are useful in arriving at the quantity of flow by sum or difference.

In rock tunnels or other such type air courses, there may be a section several hundred feet long in which the measuring station may be located. The available length of air course should be visually inspected, and the most promising spot selected. Customarily some one spot will stand out as a best location.

After having selected and marked stations in two different mines, I had occasion three years later to resurvey the ventilation. In most places the old paint or chalk marks which I had placed on the ribs were laden with dust, and were not discernible to one passing by. In selecting the new stations, however, invariably the same site would be chosen as had been used before, as evidenced by discovering the old station markings along side the new ones.

Great care must be exercised when locating stations in timbered air courses, because at a timber set the blocking between cap and roof, legs and ribs may not be closely spaced. Considerable air may be passing through these openings, and, of course, this air flow would not be

Questionnaire and Statement Which Formed the Basis for This Paper

QUESTIONS ON USE OF ANEMOMETERS

1. Has your company or agency issued written instructions for care and use of anemometers?__________ If so, will you please enclose a copy with your reply.

2. Are your anemometers calibrated periodically?__________
   a. If so, by their manufacturer?__________
   or by?__________
   b. Are calibration corrections applied to all observed mean velocity readings? __________

3. Do your men hold the anemometer by hand in measuring air flow?__________
   a. If so, do they hold it in one central position?__________
   for 1 minute?_____or traverse the section?_____for 1 min?_____
   or at?_____points for 5 seconds each?_____

4. If the anemometer is not held by hand, how is it supported?__________

5. Is each initial determination repeated immediately after it is taken as a check? __________
   a. If so, can you supply data on the self-consistency of such measurements, i. e., how well do the repeat runs check the first runs?__________

6. Has your company or agency issued instructions for choice of measuring section, covering such points as desirable approach and departure conditions, freedom from obstructions, preferable shapes and dimensions and methods of determining cross-sectional area?__________
   If so, will you please enclose a copy with reply.

7. Do your men take air measurements in or near regulator openings?__________
   a. If so, directly in the opening or how far downstream from it?__________

8. Do your men accept measuring sections which are partly obstructed by a prop or props in the section or within a foot or two of it?__________
   a. If so, do they deduct the area or projected area of the prop or props in computing net area of the section?__________

9. How are sections measured to determine their areas?__________
   By tape only__________; by plumb bob and tape__________; or by special instrument or methods__________? If special methods are used, please describe__________

10. General remarks__________

measured in the anemometer traverse. Smoke apparatus may be advantageously employed to disclose such conditions.

When a station is selected in an untimbered location, vertical lines opposite each other are placed on the ribs to guide the operator during the traverse.

The cross-sectional area of the station may be determined by using a tape graduated in feet and tenths of a foot. The degree of regularity of the section contour will determine the number of measurements to be taken. For most mine openings, one or two vertical measurements and from 2 to 6 horizontal ones will suffice.

The station should be traversed, and a repeat measurement made.

The man running the traverse may hold a watch in his free hand, or a second man may stand close by the rib, well downstream, and call out the time.

The ability of an operator to move the anemometer at a steady rate and to form a pattern so as to cover adequately the entire cross-sectional area, and complete the traverse on time, after covering, once or twice, the entire section, is a skill which can come only with continued practice.

If the station has sufficient height, the operator may pivot with the anemometer overhead while midway through the traverse, in order to cover the entire section with the instrument and complete the traverse with the anemometer along the second rib. Otherwise, he should start at one rib and end the traverse with the anemometer close by himself as he backs up against the second rib, then reverse sides, and average the results.

The operator can easily hold the anemometer so as to start and stop its recording mechanism with the index finger of the same hand which holds the machine. In spite of this, many men will fumble around when starting and stopping the instrument, disturbing the air flow and getting 65 sec or so of recording into what is supposed to be 60 sec worth.

Probably the greatest abuse, however, is that the operator, rather than passing the instrument through a definite traverse pattern, will whisk the anemometer about haphazardly in the central part of the station. Such practice obviously cannot result in obtaining accurate air measurements.

The velocity recorded on the anemometer must be adjusted to a true value by using the calibration correction for the instrument.

A "method" factor must be used also to compensate mainly for distortion of the recording through hand-holding instrument. By reducing the full measured cross-sectional area by 4 sq ft, a simple and suitable arrangement has been established which will take care of the method factor for all standard conditions.

Some measurements must be made in very small openings, in inclined openings, or in locations having other poor natural

December 6, 1947

UNDERGROUND ANEMOMETRY

BY CLOYD M. SMITH

Description of two methods of using the rotating vane anemometer for measuring rates of air flow in mines and proposal for investigations.

Submitted to members of Ventilation Committee, Coal Division, AIME, and other interested parties for discussion.

The rotating vane anemometer has been the principal instrument for measuring rates of air flow in mines since the earliest days of sytematic ventilation, and it has won an indispensable role in that field. However, it is one of the most abused of engineering instruments, and a shamefully small amount of thought has been devoted to its use. The principal abuse of the anemometer lies in its being held in the hand during registration, a method which brings the operator's body so close to the measuring section and to the instrument as to disturb the flow of air through the section and past the instrument and thereby to distort the results. Whether the hand-held anemometer is held in one position or traversed about the section, there is sufficient evidence to rule out this method of air measurement for any but the roughest estimates of rates of flow. Nevertheless, the anemometer is held by hand by company, state and federal inspectors in thousands of mines daily.

If accurate results are to be had, the anemometer must be supported mechanically and the measuring section must be traversed. Traversing is usually done by holding the anemometer for 5 sec in each of several well-distributed positions in the measuring section. Two methods of mechanically supporting the anemometer in the section during traversing are in use. In one the instrument is hung from the end of a long stick or rod by a hasp at the top of the anemometer frame in such a way that the instrument's axis of rotation and the supporting rod are in a vertical plane parallel to the direction of flow. In traversing, the operator moves the instrument from point to point in the measuring section, while standing well downstream from the section. As nothing but the anemometer is introduced into the measuring section this method will be referred to as the clear section of traversing.

In the other method the instrument is rigidly attached to the end of a long stick or rod by a screw socket on the cylindrical frame of the anemometer in such a way that the rod is perpendicular to the axis of rotation. In traversing, the operator stands centrally in the passage close enough to the measuring section to hold the rod and instrument in the section and to move the anemometer from point to point without shifting his feet and with minimum bodily movement. Inasmuch as he must essentially stand in the section, facing the air stream, this will be referred to as the obstructed section method.

Both methods have advantages and disadvantages. The clear section method leaves the pattern of flow through the measuring section unaffected if the operator stands a sufficient distance downstream from the section. This reduces to a minimum acceleration and deceleration of the rotating parts of the anemometer due to changes in air speed from point to point. This is conducive to accuracy. An objection to the clear section method is that the operator stands directly downstream from the instrument so that failure to have a support long enough to remove the upstream effect of the operator's body from the traverse section affects the registration. The error is minimized by lengthening the support but requirements of convenience limit its length to a little more than 6 ft. A theoretical objection to the clear section method is that the operator moves from one side of the passage to the other in traversing. In doing so he gets into zones of different velocity, and presumably sets up differing resistances to flow which could affect the rate of flow. This effect is presumably negligible, but it is subject to investigation.

The obstructed section method has the theoretical advantage of maintaining constant flow within the system during traversing but it has the disadvantage of setting up an abnormal flow pattern in the traverse section. The abnormality comes about from the fact that the operator must stand so close to the section that he obstructs flow in the part of the section that is in front of his body. This speeds up flow else-

where in the section and sets up steep velocity gradients between the area directly in front of him and the adjoining portions of the section. Such gradients call for a finer network of traverse points than would otherwise be required and they lead to rapid acceleration and deceleration of rotating parts of the anemometer when it is moved between obstructed and clear points. Rapid changes in speed induce instrumental error and impair application of calibration corrections.

The obstruction of flow in front of the operator may be such that the vanes of the anemometer come to a full stop while the instrument is stationary during 5-sec period. It would seem that such an abnormal flow pattern would make it impossible to get accurate results, but apparently the method can be made to yield acceptable mean velocities.

In both methods the operator must leave his normal position at the beginning of a traverse to start anemometer registration and at the end to stop it, but this requires only 2 or 3 sec at each end of the traverse so the percentage of total time involved is small. It is probable that neither method is better than the other in this respect.

The two methods should be proved against rates of flow determined by Pitot tube traversing. To do this, underground measuring stations should be established in which rates of flow could be measured by Pitot tube and by the two methods of anemometer traversing at different rates of flow. A range in mean velocity from 100 to 2000 fpm would be ample. The tests should also be run in sections of different size and shape, ranging from 5 by 7 to 10 by 12 ft in cross section. Comparison of tests run with calibrated anemometers by several observers using both methods should lead to conclusions as to superiority of one method over the other from the standpoints of accuracy, consistency of results, and adaptability to use by miscellaneous operators.

ahead of me due to the presence of my body in the air stream.

My suggestion is why not combine advantages of both methods by conducting the clear section method with a stick sufficiently long to permit the operator to remain stationary throughout the traverse. This method should satisfy everybody, and from past experience I am convinced that the results so obtained should be in sufficiently close agreement with the results by the so-called "obstructed method" as to justify the use of this more wieldy practical method.

In summation, it has been my experience that the anemometer's accelerations and decelerations occasioned by body presence at arm's length below the instrument are of the same order of magnitude as instrument accelerations and decelerations experienced by the normal flow pattern encountered in the average mine airway where its measurements are made. For example, the instrument slows down considerably more upon approaching close to the roof or rib than it does when passing in front of the operator's body. And, furthermore, the stationary operator guarantees a steady flow pattern throughout the entire traverse, which I think outweighs the effects of the anemometer's accelerations and decelerations to which you refer.

characteristics. It would be extremely difficult to attempt to set up definite procedures for such cases, and the individual must rely on his judgment as to how he will proceed, and must also evaluate, on the basis of his experience, the validity of measurements so obtained.

It is well to mention here, too, that too frequently anemometers are used to measure low-velocity air currents to which they are unsuited. Such low-velocity currents can be measured suitably with smoke apparatus.

Unquestionably there is much to be learned concerning the best methods for using anemometers, and tests, under controlled conditions with the various methods available, would certainly bring out much valuable information.

STEPHEN KRICKOVIC—I believe your subject matter should be clarified from the standpoint of job to be done. If a fan test is to be run or if total volume entering or leaving the mine is desired, greater accuracy is needed than in measuring air volumes throughout the mine. For the latter, I have found traversing of an area twice per minute at a uniform rate of speed, but without marking off the area into a number of uniform squares, with one or two immediate checks is perfectly satisfactory. Actually, I have obtained good results by the same method on fan tests. Generally, however, fan test volumes, if not obtained by Pitot tube, should be measured in the divided areas.

It is to be noted in all of this work that an accuracy of 5 to 7 pct is generally satisfactory.

RAYMOND MANCHA—The following is my appraisal of the situation expressed in comparatively few words.

Body effect upon flow pattern extends considerably more than 6 ft upstream resulting in a continual shifting or changing flow pattern in the plane of traverse when employing what you describe as the "clear section" method of traversing with a constantly moving operator. This effect can be easily determined quantitatively by the application of existing formulas from the effect of obstructions in an air stream at various distances from said obstructions.

Complete or near stoppage of the anemometer held directly ahead of the "obstructed method" stationary operator, is entirely foreign to my entire range of practical experience. I have never seen an anemometer stop or even approximately stop when held at an arm's length

C. H. MATTHEWS—Once each month our section foremen check the flow of air at various pre-measured locations and we send a report to the state mine inspectors. If the flow of air does not correspond closely with previous readings a check is made by one of our mining engineers. Anemometers are sent to the manufacturer for calibration, but we have often found that the foremen do not always use the correction factor. The periodic tests made by our section foremen are really just an indication of air flow from month to month and errors may be in all of their readings.

When a new fan is contemplated, a fairly careful check is made to determine the pressure required for the flow of air requested by the mining department. The only need for great accuracy might be to check the performance guarantee of a fan. This is not as necessary today as it was 20 yr ago because the manufacturers have modern facilities for testing and can easily meet their guarantees.

We endeavor to provide adequate air where needed and to install fans that do not waste power.

R. E. CONDON—Referring to your letter and to your very emphatic statement

that holding an anemometer in your hand is "no good," I cannot entirely agree with you in this statement, as I believe anemometer readings made in such a manner are very good for quickly determining the relative difference in volumes, or rather velocities from day to day, which actually is what the inside personnel of the mine are interested in. It tells them very quickly whether any unusual condition has come about in the short period of time since the last measurement was taken.

I do agree with you in that it is not an accurate way of determining the actual volume or velocity that is passing that particular point. I also believe that the areas measured at the measuring station lend as much inaccuracy to the readings as the method of taking anemometer readings.

In discussing your statement on "Underground Anemometry," I would say first that the same condition would apply when holding the anemometer in your hand as mentioned above. The error which comes into such readings varies considerably with the velocities that are being measured and, in my experience over many years, indicates that on velocities around 100 to 200 fpm, which are usually encountered near the working faces and with the large areas that are involved, the difference between the readings obtained in the hand method of taking readings and those obtained by mechanically supporting the anemometer, is very small. By this statement I mean the different readings would not vary more than 5 pct, which, of course, is still in favor of the mechanically supported anemometer.

Whereas, in velocities exceeding 1500 fpm, this same difference would vary as much as 30 to 40 pct in the same direction. In general, it is undebatable that the more accurate readings can be obtained with the mechanically supported anemometer.

In regard to the two different methods of supporting the anemometer for taking measurements, it is my experience that the one which you mention as being the obstructed section method, is the more practical one for everyday use and I believe is just as accurate as the clear section of traversing. Explaining further this statement, I find that very few mine operators are concerned enough about the accurate readings to such an extent that they will provide an ideal measuring station underground, and the net result is that one has to find the best location under existing conditions to obtain the readings. A good many factors must be considered when trying to locate such a station and I am happy to say that the recent trend brought about by the Department of Mines in specifying the fan to be offset a minimum distance of 15 ft from the near edge of the mine opening, has in a good many cases presented a better measuring station for air measurements, insofar as the actual volumes passing through the fan are concerned. This means there is sufficient space immediately upstream from the fan to take a volume reading, which would include practically all leakage air present in the ventilation system; whereas, if the readings were taken underground there is a possibility of a large leakage occurring between the point of measurement and the fan.

This same underground reading, however, without a doubt would be much more accurate in measuring the volume of air passing through the mine itself, providing usual precautions are taken when taking the measurements.

You have stated the common errors prevailing in using both of these systems and in either one extreme care must be taken in order to accomplish most accurate results. For the inexperienced operator I would suggest stretching strings across the area, which are used as guides when taking measurements. An experienced observer can take such readings either by using marks along the sides of the air course, or simply by using other marks of identification usually present in an air course.

It is my opinion that either of the methods stated should check very closely on successive readings and I believe should come within 2 pct, even on the higher velocities. I do believe that successive readings should be taken with the mine in operation, as local conditions quite frequently vary the readings considerably during the time interval required to make the traverse.

I believe your analysis and description of the two methods is very complete and I cannot think of anything more to add. I do believe, however, that there would be little gained as suggested in the last paragraph of your report, by establishing an underground station and testing the two different methods of taking the readings, because it might be misleading. In other words, there is no doubt in my mind that with such a perfect setup, both methods with experienced operators, would check very closely; whereas, I can think of a number of places under practical conditions, wherein one method would be better than another and it is my contention that wherever readings are taken, extreme care in handling instruments, taking measurements, and applying good common sense, are necessary to obtain accurate results.

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Leo Glanville—I definitely agree with you that the velocity readings obtained by an anemometer held in the hand are erratic and erroneous, due to the obstruction of the operator's body in the air stream. The smaller the cross section of an airway and the higher the velocity, the greater the error.

Occasionally where a grab air velocity reading is taken and accuracy is not required one center reading is taken and a correction factor of 0.8 is used.

On monthly routine ventilation surveys a 1-min traverse (12 positions of 5-sec each) is made and then immediately repeated, reversing the sequence of positions. The average of the two readings, after correction by calibration chart, is accepted as the mean velocity.

When an anemometer is used to determine fan efficiencies the same procedure is followed, with the exception that at least 20 positions of 5 sec each are used.

The clear section method is always used, with the anemometer held at least 5 ft in front of the operator.

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D. A. Harrington—As to the use of the anemometer, you are correct in calling attention to the gross carelessness in almost every phase of the use of the instrument in practically all mines and by practically all persons in all mines. I have realized for 30 yr or more that anemometer air measurements of the usual kind, by holding the instrument in the hand usually with the body in the air stream, are mere approximations and this is true to even a greater extent of the usual tape measuring of the cross sectional area.

These two *guesses* are multiplied together and the quantity thus secured is at best a not particularly close approximation as to the actual quantity available. However, generally this does not bother me inasmuch as very seldom, in my opinion, are extremely accurate measurements necessary except possibly when a ventilation survey is made to determine the size, type, and so on, of fan to be installed or the excess in power cost due to defective airways, overcasts, and other causes. Not 10 pct (probably much less than 5 pct) of those who use anemometers have the education to realize how to do accurate work with an anemometer or to obtain accurate results from the measurements or even to have anything like an adequate idea as to the significance of the measurements or the results (or expected results) obtained or obtainable from them.

In general anemometer work in mines, especially coal mines, is done largely to comply with some "*must*" law or regulation and when that is done responsibility ceases.

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Carl Lee—Commenting on your paper and its relation to ventilation prob-

lesm in our coal mines, I believe it can be definitely stated that we have two separate and distinct problems. The first of these is that of the mine managers, face bosses, safety men, and others, who take readings of the volume of air on the various splits and at the faces.

It has long been standard practice that this is taken with a hand-held anemometer in almost any convenient place, without too much technical refinement for accuracy. It is reasonably certain to state that usually the volume of air as calculated from such measurements is higher than the actual volume if determined by more careful and possibly scientific measurements.

I believe that the majority of the coal companies, as represented by the officials thereof, will be willing to let this practice continue as is.

The second consideration in the measurement of air is that of accurately determining the volume of air in the various splits so as to coordinate with pressure surveys for the purpose of: (1) calculations of possible changes in the coursing of air to improve same, or (2) with respect to the performance of the existing fan or proposed replacement fan.

For the purpose of getting accurate readings, it certainly is desirable to use some refinements in measurement of air, so that the total cfm will reasonably check with the performance curves of fans as offered by the manufacturers. It is also important to have reasonably accurate volumetric reading in making estimates of recoursing of air for improvement.

However, with our company for example, it has been the writer's opinion that a pressure and volumetric survey of a mine rarely warrants the time required to calculate carefully all of the small corrections at every point, for air temperature, velocity head, and so on. Put in other words: at present, we consider it more important to get approximate surveys of all of our mines, rather than too refined tests on laboratory accuracy standard of only a few mines.

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Richard Maize—The rotating vane anemometer is a very practical instrument and is used by mine officials, safety engineers, and mine inspectors to find out if the fan is producing sufficient air to meet the requirements of the state law and whether the volume is sufficient at the working faces to dilute and render harmless all explosive and noxious gases. As a general rule, those using an anemometer for this purpose do not make the calibrated corrections, nor do they make several readings at different points in the cross-sectional area in order that an average mean velocity may be obtained.

In measuring the volume of air necessary to meet the requirements of the law, it is not necessary, in my opinion, to sectionalize the airway and hold the anemometer on a rod in each imaginary section. If the operator stands in the airway in such a position that the anemometer will be held in the approximate center of the flow of air, the operator's body will cause some turbulence in the air as it flows through the anemometer. However, for all practical purposes, this interference should make very little difference when all subsequent readings are made under like circumstances. If one is going to be so particular as to sectionalize the airway, he should then make some allowance for the area of the airway which is obstructed by his body, and deduct this from the cross section of the heading where the measurement is being taken.

When an operator is taking a reading in a regulator to determine the amount of air that is entering or leaving any particular section of the mine, no allowance is made for the "vena contracta." If the velocity of the air entering a regulator is high, the measurements are taken far enough away from it so that the readings are not affected by the high velocity of the air as it is rushing through the regulator. If there is a small volume of air entering the split, the operator usually holds the anemometer in the regulator, and in this case, the only obstruction encountered by the air as it passes through the regulator is that caused by the anemometer and the operator's arm.

Precautions, such as sectionalizing the airway for velocity measurements and the making of calibrated corrections, are all very necessary when determining the efficiency of a fan. The humidity of the air is another important factor and will greatly affect accuracy if not considered where large volumes of air are involved. In practice, it is possible that there be a difference of more than 2000 lb in the weight of the air delivered by the fan each minute, and a greater or less horsepower will be required to maintain the same fan speed, depending upon the relative humidity of the air.

A difference in temperature between the intake and return airways is another factor which tends to affect greatly the efficiency of a mine ventilating fan. It is not uncommon to find 20 pct less air entering a deep mine in the summer than that entering during the winter season, all factors being equal except the temperature.

My conclusions in reference to this problem are that the anemometer is not a badly abused engineering instrument. It will give the information one is seeking, whether for practical purposes, as in the normal operation of a mine where air measurements are made by the foremen as required by law, or when one is seeking technical information relative to fan efficiencies.

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A. S. Richardson—I entirely agree with you that holding an anemometer in the hand is no good if the body is in the air stream. We make all our measurements by what you call the clear section traverse, using a number of fixed points central in equal areas, with equal periods of time at each area. However, we move from one point to another without stopping, and starting, and the traverse is made so that the anemometer can be started and stopped by the operator without it being necessary for him to move from his fixed position, out of the air stream. Also, I consider it to be essential that the reading be made near the center of an undisturbed straight run of at least 100 ft.

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G. E. McElroy—I am opposed to the "clear-section," or downstream method of measurement because it is an awkward method and because I believe the air distribution is disturbed at a distance of 6 ft upstream from a man's body even though photographs of air-streams do not indicate that this would be the case. I have in mind a particular case where Pitot-tube velocity pressures were seriously affected by a 4-in. wide obstruction 4 ft downstream.

For ordinary mine air measurements I advocate holding the anemometer by hand in a way that offers minimum interference and considering the reading so secured as 5 pct high.

References

1. G. E. McElroy and A. S. Richardson: Resistance of Metal Mine Airways. U. S. Bur. of Mines *Bull.* 261 (1927) 103–110.

2. G. E. McElroy: Engineering Factors in the Ventilation of Metal Mines. U. S. Bur. of Mines *Bull.* 385 (1935) 6–14.

3. Pages 33–36 of ref. 1.

4. Callen and Smith: The Measurement of Air Quantities and Energy Losses in Mine Entries. Univ. of Ill., Engrg. Exp. Station, *Bull.* 158, Part 1, 14–16.

5. Pages 45–49 of ref. 4 and also *Bull.* 170, Part 2, 28–33.

6. Raymond Mancha: Ventilation. *Mechannual* (1945) 182–183; and The Jeffrey Mfg. Co., *Catalog* 797 (1946) 40–41.

Discussion

B. F. Tillson*—The manifold difficulties of accurate anemometry in irregular sections of mine passageways, the irregular distributions of velocities in cross sections of the same, and the disturbing influence of the observer upon the air flow, all indicate an undesirable inaccuracy of results obtainable by any standardized method of traversing the section by an anemometer.

It seems obvious that another, and simpler, method should be used to determine the volume of air flow in mine passages, namely:

1. At appropriate locations cement or calked framework rings should be installed permanently to equalize the irregularities of sectional contour and provide a place and means of attachment for a temporary cloth brattice which bears a rigid orifice.

2. The measurement of the velocity of air flow through the orifice may then be by anemometer or, preferably, by Pitot tube measurements of the differential pressures on both sides of the orifice in accordance with the standard practices available in engineering literature.†

The constants may be determined for various measuring positions in relation to the resulting "vena contracta."

3. The position of the person who makes the measurements is behind the brattice out of the air stream. The Pitot tube does not offer as disturbing an obstruction as the anemometer. A recording gauge may be employed to integrate fluctuations in air flow through that portion of the mine. No traverses are required because the reading may be at a single central point. An anemometer can be used with an orifice flow. The orifice will increase the air velocity at the measuring point, with correspondingly more accurate measurements where the normal air velocity through the passageway is low. Portable brattices might be devised with the cushioning rims which would seal against irregular rock surfaces where permanent rings were not available or feasible.

The development by the Ventilation Committee of standard procedures and devices for the orifice measurement of the flow of mine ventilating air might be a desirable project for this coming year.

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† *Trans.* A.S.M.E.

C. M. Smith (author's reply)—Thank you for your discussion of my paper on underground anemometry. Your suggested method of measuring underground air flow is a novel one which might be applicable in some situations.

It should be tested along with other suggested methods in any investigation of this subject.

G. E. McElroy*—In spite of the adverse publicity that vane-anemometer methods of air measurement have had in the past and that contributed by the present paper, I endorse Mr. Krickovic's statement that anemometer traversing "has proved to be widely applicable, expeditious and simple" and add that available methods are accurate enough for the purposes for which they may be used.

The fact that the great majority of minor mine officials assess relative changes in rates of air flow by comparison of crude vane-anemometer measurements, known to average 20 to 30 pct high, has no important bearing on this subject, because state inspection standards were based originally on such methods of air measurement. Federal inspection standards are based on actual rates of flow as determined by traversing, and interest in traversing methods is rapidly increasing.

In considering traversing methods, three aspects are of major importance: (1) the absolute accuracy of calibrations; (2) the degree of interference with normal flow conditions introduced by traversing methods designed for accurate measurement by shaft-mounted instruments; and (3) the proper "method" factor to use for approximate measurements by hand-held instruments.

With respect to absolute accuracy of calibrations, we have always placed reliance on calibrations made by the National Bureau of Standards, with which manufacturers' calibrations have usually agreed very closely. It is therefore particularly disturbing to find[7] that calibrations made previous to June 1947 are presumably about 5 pct in error because of excessive registry caused by the thin flat plates on which anemometers were mounted for calibration. Velocities corrected for calibration have therefore averaged about 5 pct low in all probability. In this connection, it is interesting to note that an anemometer calibrated against Pitot-tube measurement by a single-point method in the Bureau of Mines experimental coal mine in 1923 indicated this same difference of about 5 pct and that the same instrument calibrated by a traversing method in a metal mine some months later indicated a difference in the same direction of about 4 pct. These results are reported by the Bureau of Mines.[8]

Regarding the degree of interference, or changes in velocity distribution, caused by the position of the observer's body in traversing operations, misconceptions seem to be especially prevalent, resulting in increasing advocacy of methods, such as the "clear section" method outlined in this paper, that cause just the type of interferences that they are designed to avoid. The degree of interference for any method may be gauged easily by a few experiments with a velocity-pressure gauge connected to a Pitot tube or with an indicating velocity meter such as the Velometer. In an experiment cited by McElroy and Richardson,[9] a decrease of 5 pct was noted at ten widths upstream from a 6-in. plank, whereas an observer's body at about the maximum practical distance of 6 ft downstream from the instrument is only about four widths away. In the Bureau of Standards paper previously mentioned, it is recommended that supports used in calibrations be at least 16 widths downstream. In practice, therefore, a downstream position of the observer is ruled out as far as accurate measurement is concerned.

Operation of the anemometer by rigid shaft support from a point outside the section is seldom practicable; however, accurate results can be obtained, with the anemometer rigidly attached to a short shaft and held at arm's length, by an observer advancing across the traversed section while he faces the opposite wall and stands sideways to the current, *provided* that he keeps the instrument at least 3 ft away from his body at all times and traverses the entire section with it. If the traverse can be started with the observer in a side recess, the entire section can be covered in one operation. Normally, it would be covered in two half-sections. The presence of the observer's body does not, as is commonly supposed, increase the average velocity throughout the remaining part of the section. Rather, the velocities 1 to 2 ft on either side of his body are increased, but the distribution of velocities throughout the rest of the cross section remains normal, and a traverse made as stated gives a true average velocity for normal-flow conditions.

Regarding the proper "method" factor for accounting for interference in the approximate methods of traversing with hand-held instruments, here again confusion prevails, for which the writer must assume some of the blame. Comparison of consecutive traverses made by shaft-held and hand-held 4-in. anemometers in field work after the tests reported by McElroy and Richardson[1] gave method factors

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[7] Galen B. Schubauer and Gerald H. Adams: Effect of Support on the Performance of Vane Anemometers. National Bureau of Standards Research Paper 1872, *Jnl of Research.* (April 1948), 40 275–280.

[8] Page 104 of ref. 1.
[9] Page 102 of ref. 1.

averaging about 0.90 in 40 to 50 sq ft metal-mine airways but averaged slightly higher—about 0.92—in a few 70 to 90 sq ft sections. Considering the effect to be one of relative size of airway, the figure was increased to 0.95 for coal mine airways and was so reported to an early class of Federal coal-mine inspectors and in answer to Mr. Smith's questionnaire. Apparently, the increase of factor from 0.90, as supported by experiment, to 0.95 based on opinion, was unwarranted. The figure of 0.90 is independent of the calibration, as both readings were corrected in each experiment before the ratio of 0.90 was determined.

Analyses of average results of the four most reliable groups of similar 4-in. anemometer traverses reported by Callan and Smith,[5] by comparison of calibration-corrected anemometer averages against Pitot-tube averages, give factors of 0.86, 0.85, 0.82, and 0.82 for Sections A, N_2, N_4, and Q_4 respectively. As the Bureau of Standards calibrations[7] then used were presumably about 5 pct high, these figures should be divided by 1.05 to obtain the corresponding method factors independent of calibration. These are 0.82, 0.81, 0.78, and 0.78, respectively.

Recent wind-tunnel tests by the Bureau of Standards[7] on the effect of supports on anemometer calibrations gave results averaging about 17 pct high for a 4-in. anemometer with two artificial-hand supports somewhat resembling those used in practice. The corresponding method factor, which does not, of course, take traversing procedure into account, is 1/1.17 or 0.85. Actually, the results of the tests on the 4-in. anemometer are somewhat out of line with comparable tests of the same series on 3 and 6-in. anemometers, and a possible factor of about 0.87 is indicated.

Because it is so sensitive to minor differences in the manner of holding the instrument, hand-held traversing should be used only for rough approximations. A suitable method factor for the exact method of holding can be determined preferably by consecutive traverses in a steady air current made alternately with the vane anemometer shaft-mounted and hand-held.

In my opinion, no elaborate program of tests of anemometer traversing methods is needed except such as may be required to convince those contrary-minded that no method should be used that involves placing the observer downstream of any traverse position or within 3 ft cross stream from any traverse position.

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C. M. Smith (author's reply)—It seems probable that the flow is so disturbed in all of the proposed methods of traversing as to lead to errors in the results; on the plus side in 2-position method (see Table 1), on the minus side in the multiple position and erratically on either side in the 1-position method. We will be in the dark as to the nature and extent of the errors until we test all proposed methods of anemometer traversing against some independent method such as Pitot tube traversing. I believe that underground anemometry is so important to health and safety in mining as to warrant the time and expense involved in putting it on a sound engineering basis.

Importance and Application of Piezoelectric Minerals

BY HUGH H. WAESCHE* Member AIME

Of all the military services, the Signal Corps is the most concerned with piezoelectric minerals because of its function as a supply service to the strategic and tactical military forces. Consequently this paper is written from the point of view of one associated with that organization. The Signal Corps is responsible for the research, development, and supply of communications, radar, and components to the using services of the Department of the Army and to some extent the other branches of the National Defense Department. Their work therefore includes the study of the sources, characteristics, and application of quartz and other piezoelectric materials. These materials have become a vital consideration in strategic planning and are essential for efficient tactical operation by all the Armed Forces. The Signal Corps at the beginning of World War II was responsible for both Army Ground and Air Force electronic equipment. Since that time this Army service organization has probably done more in the development of frequency control devices using piezoelectric materials than any other group.

The U.S. Department of the Interior, Bureau of Mines, Minerals Yearbook of 1945, shows that during the four war years, 1942 through 1945, 9,598,410 lb of quartz crystal were imported for all uses and of this total, 5,168,000 lb were consumed to produce 78,320,000 crystal units for electronic application. Other government records confirm these data which conclusively show that approximately 53 pct of the crystalline quartz imported was consumed in the production of electronically applied quartz crystal units. It may be assumed that some effort was made to maintain a stockpile over demands for all purposes, and this would mean that the actual percentage of quartz used electronically was considerably over the 53 pct figure. These data only emphasize that electronic application of crystalline quartz was the greatest requirement, and perhaps the actual value in this application to national defense is many times greater in importance than is apparent on first inspection. Current electronic research and development programs of the Armed Forces are planned around the fundamental use of piezoelectric minerals for frequency control and this at present, at least, means quartz.

Definition and Early Development

The word piezoelectricity is formed from combination of the Greek word "piezein" meaning "to press," and "electricity." It is that property shown by numerous crystalline substances whereby electrical charges of equal and opposite value are produced on certain surfaces when the crystal is subjected to mechanical stress. It appears to be intimately associated with the better known property, pyroelectricity and in fact, the two may be manifestations of the same phenomenon. This property was discovered by Pierre and Jacques Curie in quartz, tourmaline, and other minerals in 1880 while studying the symmetry of crystals. The converse effect, that is, mechanical strain in the crystal when placed in an electrical field, was predicted by the French physicist, G. Lippman, in 1881, and verified by the Curies almost immediately. As has been the case with many discoveries of similar character in the basic sciences, not much attention was paid to this property for many years except as an entertaining curiosity. Between 1890 and 1892 a series of papers was published by W. Voigt in which the theoretical physical properties were put into mathematical form. The first practical application of piezoelectricity occurred during World War I when Professor P. Langevin of France used quartz mosaics to produce underwater sound waves. The same mosaics were used to pick up the sound reflections from submerged objects which were in turn, amplified by electronic means and used to determine the distances to such objects. This device was intended for use as a submarine detector but development was not completed in time for war service although it was used later for determining ocean depths. About the same time, A. M. Nicholson, of Bell Telephone Laboratories, developed microphones and phonograph pickups using Rochelle salt crystals. A major step in the application of piezoelectric quartz came in 1921, when Professor W. G. Cady, of Wesleyan University, showed that a radio oscillator could be controlled by a quartz crystal; from that date, this use of quartz has increased steadily, reaching its peak in World War II as is shown by the figures previously given. Essentially all American electronic equipment for communication, navigation, and radar, utilized quartz crystals for the exacting frequency control required.

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Crystalline Minerals with Piezoelectric Properties

QUARTZ

Hundreds of piezoelectric crystalline materials are known, most of which are water soluble. Of these, quartz appears to be without a peer for electronic frequency control. Unfortunately, the quartz must be of superior quality. It must be free of mechanical flaws, essentially optically clear, free of both Brazil and Dauphiné twinning and must be, for average uses, over 100 g in weight. Because of these stringent requirements, raw quartz of the quality desired is of rare occurrence. In addition to quartz, several other naturally occurring crystalline materials are known to have the piezoelectric property and could perhaps be substituted for quartz in some applications. These

include tourmaline, nepheline and berlinite.

TOURMALINE

Tourmaline has certain desirable properties which would assure its widespread use if available in suitable quantity and quality; only light colored (noniron bearing) varieties are suitable for frequency control. Tourmaline of this type, of large size and in flawless crystals, is even more rare than suitable crystalline quartz. No known slice of tourmaline, cut with specific relations to the crystallographic axes, has the so-called zero-coefficient frequency-temperature characteristics of certain quartz "cuts";* it could not be used, therefore, in most of the precise frequency control applications that are normally required in electronics. However, tourmaline has a much higher frequency-thickness constant which makes it possible to attain fundamental frequencies higher than those of quartz. It was reported that the Germans produced during the war some tourmaline crystal units that had fundamental frequencies of about 400 megacycles per second. Tourmaline at this frequency is so thin—around 6 ten-thousandths of an inch—that it is almost invisible. A slice of quartz having this frequency would be 4 ten-thousandths of an inch thick when cut for highest frequency-thickness ratio. A thickness of this value is impractical; in fact quartz is not economically handled in current production if the thickness becomes much less than 4 thousandths of an inch for the preferred cuts.

NEPHELINE

Nepheline is assumed, on a theoretical basis, to be piezoelectric although its characteristics have not been measured, but qualitative tests have shown a positive piezoelectric effect. No known naturally-formed nepheline is suitable for piezoelectric use because the crystals are too imperfect, too small, or lack crystalline form as is true of quartz occurring in igneous rocks.

BERLINITE

The aluminum phosphate mineral, berlinite, is piezoelectric, based on theoretical and practical evidence. However, no known natural berlinite crystals are large enough for any practical use. The general crystalline structure of berlinite appears to be identical to that of quartz except the *c* axis of the unit cell is twice that of quartz. These four minerals are the only natural ones now considered actually or potentially applicable to piezoelectric use.

SYNTHETIC SUBSTANCES

A long list of artificial water-soluble crystalline piezoelectric substances has been investigated. Of these, only a few are of practical importance such as Rochelle salt ($KNaC_4H_4O_6 \cdot H_2O$), ADP (ammonium dihydrogen phosphate $[NH_4]_2H_2PO_4$) and recently, EDT (ethylene diamine tartrate C_6H_{14}-NO). The latter was described by W. P. Mason,[1] of Bell Telephone Laboratories. Mason also suggests that DKT (di-potassium tartrate $K_2C_4H_4O_6$–$\frac{1}{2}H_2O$) may be a promising piezoelectric material. Of more than casual interest are some of the other tartrates and various borates, titanates, and sulphates. Although these crystalline substances have many and important applications, none have so far been useful for frequency control where any degree of stability or accuracy is required—and this remains, without question, the most important piezoelectric application.

Nature of Piezoelectric Effect

Piezoelectric theoretical considerations have been covered in great detail by W. G. Cady[2,3] and K. Van Dyke, of Wesleyan University and members of the Bell Telephone Laboratories, in publications of both book and periodical nature. The field by no means has been covered completely. However, there are certain fundamental assumptions that are generally accepted. Of the 32 possible crystal classes, 20 have the necessary properties to be piezoelectric. These classes are the ones of lower orders of symmetry. The piezoelectric effect depends upon the dipole moment of the molecule, *not* the symmetry of the crystal class. The number of piezoelectric constants in a given crystal *does* depend upon its symmetry and class. The rhombohedral-trapezohedral class of quartz is definitely in this category. The piezoelectric effect is explained by the assumption that, when a stress is applied to a crystal having no center of symmetry, the centers of gravity of the positive and negative electrical charges in the molecules are displaced in opposite directions. The displacement is proportional to the pressure, and the total charge for a given crystal is the sum of the molecular charges. In a crystal having a center of symmetry, as in the extreme case of the isometric normal class, no displacement of this nature occurs because all internal displacement conditions are equal and opposite. Conditions for resultant piezoelectric charges on a crystal apply only if it is untwinned. If the crystal is twinned, the charges produced in one twin will oppose those in the other twin, thereby reducing the overall effect. If the two portions of the twin are equal, the net charge for the whole crystal will be zero. The two types of twinning in quartz most frequently encountered are Dauphiné, known in piezoelectric work as electrical or 180° twinning, and Brazil law or optical twinning. The electrical twinning is an 180° reversal of the crystalline structure with the *c* crystallographic axis, the twinning axis, both portions having the same handness, that is, right or left in any one crystal. In the case of the Brazil law, or optical twinning, the *a* face ($11\bar{2}0$ trigonal prism face) is the twinning plane producing both right and left handed forms in the same crystal. Neither type of twinning is permissible in crystals used for frequency control purposes and of the two, small amounts of optical twinning appear to be the less detrimental. Every effort is made to prevent the use of quartz having either type of twinning in the production of crystal units. Optical twinning is readily recognized in polarized light, but electrical twinning usually can be determined only by etching the surfaces to be observed with hydrofluoric acid.

SPECIFICATIONS

At least a general knowledge of crystallography is necessary in order to produce piezoelectric working units. The crystalline elements of such units must have been carefully oriented in relation to the crystallographic structure before being cut from the crystal. It is this crystallographic orientation of the finished plate that determines certain of the most desirable per-

* The term "cut" in piezoelectric terminology refers to the angular relations between the surfaces of the slice, wafer, or bar of oscillator or filter quartz and the crystallographic axes of the crystal from which obtained.

[1] References are at the end of the paper.

formance characteristics of the unit of which the temperature frequency characteristics are the most important. The latest frequency controlling element specifications for quartz require orientations which must be accurate within at least 3 min of arc. In many cases linear dimensions are of the order of a few ten thousandths of an inch, and the frequency determining dimensions are so exacting that they are measured by frequency change rather than by the linear differences which would be of the order of subdivisions of millionths of an inch; removal of one molecular layer from the surface of a high frequency quartz oscillator plate could, under some conditions, move the frequency out of allowed tolerances. The simple appearance of a small plate of quartz ½ by ½ by 14/1000 in. belies the development, care, precision, and human energy that have gone into its production. Diamond saws, optical and X ray orientation apparatus, lapping machinery, temperature and electronic measuring and testing equipment are only a few of the items required and used in its fabrication.

Quartz

Quartz is discussed at greater length than other materials because of the unique position it occupies in the piezoelectric field. Quartz is outstanding in mechanical durability, elastic qualities that give it a very high Q (defined by Cady as ratio of stored to dissipated energy), and an atomic combination in the molecule that results in different physical properties in different directions. Because of the last factor, positive and negative temperature-frequency coefficients are available in a crystal, thereby permitting the production of so-called zero coefficient units. A zero coefficient cut oscillator plate is crystallographically oriented so that normally positive and negative temperature frequency characteristics are cancelled partially to produce a zero frequency change for a limited temperature change. This last is a property, for instance, that tourmaline lacks.

PRODUCTION AND SOURCES

At the outbreak of World War II only a handful of people in the United States had had any experience in the knowledge of or production of piezoelectric crystal units. Through a program of expansion and education sponsored by the Signal Corps and other Government agencies, the number of active manufacturers reached 150. Along with this increase in number, there was a great increase in production, and efficiency became high enough to double the yield per pound of raw quartz.

Most of the raw quartz consumed in frequency control unit production during the four war years was obtained from Brazil. Other material from Arkansas, California, and scattered deposits elsewhere in the United States was used. Some very small amounts of inferior quartz were obtained from Guatemala. It was reported that Russia had ample supplies and that sizeable deposits existed in Madagascar. The mining in Brazil was primitive and difficult and the transportation, over long, undeveloped interior distances to the coast was slow and not too reliable.

In addition to quartz, other strategic materials were required in the fabrication or composition of frequency control elements. These include industrial diamonds for sawing and grinding, silicon carbide or corundum for lapping and polishing, fluorine for etching, as well as minor quantities of beryllium-copper for springs, nickel, chromium and tin for plating, and lead, zinc, copper and silver in various other functions.

MILITARY USES

As previously stated, piezoelectric minerals from a military viewpoint are far more important than is normally indicated. Usually the importance of major equipment is widely publicized, but seldom that of the component parts. Piezoelectric minerals were given relatively little consideration before the war. It is possible to build and design certain electronic equipment that will work without quartz crystal, but the efficiency of both equipment and personnel become greatly reduced. Quick, reliable frequency change and clear channel operation require the use of crystals. No other means of attaining the same conditions are known, which explains why all military services persist in using piezoelectric frequency control devices in electronic equipment. In 1942, it was soon discovered that without a quartz crystal a radio communication set would not work, without the communication set a tank unit could not operate, and worse than that, a plane could not fly; a genuine potential case of "for want of a nail, a war was lost." Piezoelectric quartz really came into its own when bombers and fighters were grounded for lack of crystal units. It was the heart of radio communication and much of the navigation equipment.

Ground Forces

The U. S. Army Ground Forces need piezoelectric crystals principally as a frequency control element in communications equipment. Some of these required as many as 900 crystal units, although that was not the usual rule; however, 120 per equipment were not unusual. Quartz crystals were used in fixed stations, vehicular radios, walkie talkies, handie talkies, telephone and teletype equipment, facsimile equiqment, meteorological equipment; in tanks, in jeeps, on horseback, in rucksacks, in fact, in about every place a soldier or a machine of ground warfare could or would go. In nearly all cases at least two crystals were available for use per equipment at a time; one for transmitting and one for receiving. Quartz crystals were required for radar, but the quantity was insignificant compared with that needed for communication equipment. Field frequency meters and frequency standards required quartz crystals. In addition to quartz, artificial crystals of Rochelle salt and ammonium dihydrogen phosphate were used for microphones, phonograph pickups, phonograph recorders and head phones.

Air Forces

The Air Forces used many of the Ground Force equipments although to a lesser extent. The now famous "handie talkies" were a major Ground Force equipment so used. In addition, the Air Forces had several specialized requirements—all dependent on quartz crystal for frequency control. These applications were in VHF (very high frequency) command sets, requiring the bulk of the Air Force quartz crystal units, navigation equipment, and sea rescue equipment. The Air Forces procured a very large quantity of crystal units for the British Royal Air Force. The quantity production radio communication set known as the SCR-522, in every RAF and American fighter or bomber, required eight crystal units for designed operation, four for the transmitter and four for

the receiver. In addition, it was necessary to stock at least as many spares per set as well as have on hand the frequencies for over 100 channels. Use of the SCR-522 alone required the production of around twenty million crystal units.

Navy and Coast Guard

The Navy and Coast Guard demand for quartz crystals was relatively low at the outbreak of the war but increased rapidly so that by the end of the conflict they were procuring large quantities of the Air Force type of crystals plus others peculiar to their own requirements. One of the most important Navy-Coast Guard applications was in the now famous Loran equipment. This revolutionary ultra-accurate navigational aid depended on a very special type of quartz crystal unit for its phenomenal performance. The units were essentially hand tailored for each equipment and required a perfect piece of quartz crystal of medium or larger size. The Navy used large quantities of piezoelectric ammonium dihydrogen phosphate (ADP) in sonar equipment for underwater submarine detection, depth sounding, obstacle locating, and similar applications. The principle is the same as that originally developed by Langevin. Quartz, however, is no longer used since artificial crystals in this particular application are preferred because of their much greater piezoelectric effect and the frequency accuracy of frequency control applications is not required.

NONMILITARY USES

The National Bureau of Standards in Washington depends on quartz crystals for the standard frequencies that are broadcast continuously by radio. The crystals used are similar to those in the Loran equipment previously referred to except that temperature control is more rigorously maintained. This very interesting and important application of crystalline quartz is perhaps its most exacting and conclusively demonstrates the superiority of quartz in this service over any other known material. Several of the crystals operate continuously as checks against erratic behavior; however, only one is "on the air" at a time. The transmitted frequencies are continuously checked against Naval Observatory time and are guaranteed accurate to 1 part in 50,000,000 although it is believed the accuracy may be much better. This means the piece of quartz actually mechanically vibrating 100,000 times per second 24 hr per day furnishes time and frequency which are so accurate that it is now questionable if the man-made measurements of the solar system are accurate enough for confirmation purposes. It is thought possible that the quartz beat is more regular than the solar cycle, and consequently the point may have been reached where it would be difficult to say whether the apparent variations are those of solar origin or of the quartz. No tuning fork could equal this performance, nor for that matter could any other known currently available system of frequency control. The National Bureau of Standards explores all possible means and materials to this end and so far has found no substitute. The signals transmitted by the Bureau of Standards Station, WWV, are used as the standard for all the Armed Forces, commercial organizations, research laboratories, and by similar groups in foreign countries. It would be difficult to appraise the actual value in everyday living for mankind of this frequency source—depending on a piezoelectric mineral. All radio channels, broadcast and otherwise, depend on quartz crystals for accurate frequency control. It is the quartz crystal in the transmitter that makes it possible for broadcast stations to be within a few cycles of exactly the same frequency day in and day out. The accurate Loran and other navigational equipments, depending locally on quartz, in turn depend on the frequencies from the Bureau of Standards. The Bureau of Standards broadcasts the standard signals on 5, 10, 15, 20 megacycles and other frequencies.

In addition to radio, teletype, telephone, and other wire services depend on quartz or the new ethylene diamine tartrate (EDT) crystals in the transmission of numerous services over a single wire or cable. In this use, the piezoelectric material acts as a gate or filter allowing only the desired signal of the correct frequency to get through. Another widely used application of quartz is in the control of power frequencies. It is this use that makes power operated electric clocks maintain their time accuracy within a matter of seconds. This use is otherwise important to both the power suppliers and users in that meter readings are accurate and uniform. Poor regulation means the watt-hour meter runs too fast or too slow depending on the frequency. All these frequencies revert back to the National Bureau of Standards for reference and calibration. Quartz crystals are used frequently in radio communications receivers. In this case they are not detectors as was the old galena detector but instead act as filters limiting the band width of the received signal and thereby making it possible to receive more signals with less interference.

A growing use of piezoelectric materials is in the field of supersonics. In this case the piezoelectric material is excited mechanically by an electric field. The mechanically vibrating crystal transmits its energy to an adjacent or surrounding material setting up supersonic waves which travel through the medium. This is useful in homogenizing milk, making emulsions of oil in water, settling smoke, fog, or dust, and similar operations. The same principle has recently been applied to detecting defective railroad engines and car wheels. The list of uses in general, grows daily.

Toward the end of the war, many radio manufacturers considered the use of quartz crystals in home radio receivers. The idea appeared to have merit, especially for accurate push-button tuning. However, the competitive market and the additional expense have prevented such application except under special circumstances and in more expensive models. At the higher frequencies now being used more and more for frequency modulation, television and citizens radio, this application may become mandatory.

Tourmaline

Tourmaline has various excellent properties but enjoys only limited use because of the previously mentioned rare occurrence in suitable quality. It has one property of special value, lacking in quartz. It reacts to hydrostatic pressures and therefore is much in demand for blast pressure gauges. Tourmaline so far has had little application in frequency control. No other natural minerals are known to have had any piezoelectric application. Since the characteristics of nepheline have not been measured completely, its performance characteristics as yet are unknown.

Quartz Supply Program

The greatest single, deterring factor in the more widespread application of quartz crystals is the supply of raw material. As long as the source material is available and the supply lines are open, the situation may not be too critical. National defense considerations do not permit any question of supply to exist without exploring other possibilities. A situation of this nature is of prime importance to the Signal Corps which must have quartz or a suitable substitute to fulfill its mission. For this reason the Signal Corps Engineering Laboratories at Fort Monmouth, N. J., have launched a program designed to promote public interest in locating additional domestic sources (continental) for piezoelectric grade quartz and other piezoelectric minerals. This program is not intended to suggest that there are large commercial possibilities involved or that this Government Agency is anticipating large scale procurement, although it is certainly believed likely that the use of piezoelectric grade quartz would increase if larger domestic sources were developed. This program is purely for national defense consideration and is intended to be coordinated with the other concerned government organizations such as the Geological Survey and the Bureau of Mines. If commercial possibilities are opened up as a part of this program, the government can feel doubly paid for its efforts. In any case, it may be assumed that piezoelectric mineral applications will increase rather than decrease in the future and that the increase in importance both commercially and militarily will increase in proportion. Availability of larger domestic sources would undoubtedly accelerate this progressive development.

Development of Synthetics

As indicated in the January 1948 issue of *Mining and Metallurgy*, the Signal Corps is not only interested in natural piezoelectric mineral sources but it is also involved in extensive sponsorship of synthesis of piezoelectric materials. Two contracts are in effect for the synthesis of quartz. Initially this work followed the leads of Spezia and Nachen—the latter active in the efforts of Germany to produce synthetic quartz during the war. Nachen claimed laboratory production of synthetic quartz and claimed that his method would be feasible for large scale production. The investigators in this country have found it necessary to use modifications of his method and new approaches. However, all have synthesized quartz in the laboratory to a varying degree, and results so far have certainly justified the effort. The work on quartz is being done under the direction of Professor A. C. Swinnerton at Antioch College, Yellow Springs, Ohio, and under the direction of Dr. D. R. Hale at the Brush Development Co. in Cleveland, Ohio. The Naval Research Laboratories in Washington, D. C., have contributed heavily to the overall synthetic program, including quartz, in their own laboratory. In addition, at least one commercial organization is successfully synthesizing quartz independently. Tourmaline crystals of a lithium type, low in iron, but of small size have been synthesized at Baird Associates, of Cambridge, Mass., under direction of C. Frondel, of Harvard, with a Signal Corps contract. Under similar contracts, The Edward Washken Laboratory, of Cambridge, Mass., has synthesized small nepheline crystals and the Geology Department at the University of Minnesota has synthesized crystals of aluminum phosphate which may be berlinite, under the direction of Professor J. W. Gruner. Fairly large crystals of the last referenced material have been grown at the Signal Corps Engineering Laboratories, Fort Monmouth, N. J. Brush Development Co. has been growing readily, large size water-soluble crystals which are important because of potential piezoelectric use other than frequency control. In addition to the practical aspects outlined, from the academic viewpoint, the success of these projects is believed assured by contributing to the overall store of knowledge regarding crystal growth and its application to the study of ore deposits. That many of the natural minerals may have been formed at much lower temperatures than previously accepted, seems possible. As the projects proceed and more information is gained, the technical details will become available through publication and technical meetings.

Organization for Future Development

The Signal Corps Engineering Laboratories have technical personnel that accomplish or sponsor through outside contracts, the technical research and development necessary for making available the myriad components and equipment for satisfactory military communications, radar, navigation, meteorological, and associated functions. In addition to piezoelectric minerals, manganese for batteries, and mica for condensers, are outstanding among many minerals with which these laboratories are concerned. The Frequency Control Branch of Squier Signal Laboratory is the organization of these Service Laboratories responsible for development of new and improved quartz or other piezoelectric frequency control units. Geologists participate actively in the direction of planning and functioning of this Branch.

It has been suggested that the recently publicized EDT crystals would relieve the need for raw quartz. Nothing in the published literature or from other sources indicates that this is the case. Quartz remains the principal means of frequency control without any immediate hint of a real competitor. EDT crystals can and will supplant quartz in telephone resonator service thereby relieving the quartz supply problem to a small extent. This definitely extols the quality and desirability of quartz but does not correct the supply problem, especially in times of national emergency. Domestic sources of piezoelectric minerals are still the best insurance for the future coupled with a program of successful synthesis of piezoelectric frequency control crystals. Recent public announcements by both Bell Telephone Laboratories and the Signal Corps indicate successful quartz synthesis on a laboratory scale exceeding any previously reported.

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Beneficiation of Industrial Minerals by Heavy-media Separation

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THE sink-float methods designated by heavy-media separation processes were pioneered by C. Erb Weunsch for the treatment of base metal ores as an improvement over jigs. The work of Weunsch was further developed by Victor Rakowsky and The American Zinc, Lead and Smelting Co.

Early in the development of the processes, the inherent unsuitability of galena as the solid constituent of the medium was recognized and ferrous media amenable to magnetic recovery and control were developed.

The high efficiency and low cost of magnetic recovery and cleaning of ferrous media regardless of particle size, slime contamination, or surfacial oxidation had led to the adoption of ferrous media by all of the sink-float plants operating under the heavy-media separation processes patents controlled by American Zinc, Lead and Smelting Co. Approximately 2,000,000 tons of base metal and nonmetallic minerals are treated each month by these methods.

Heavy-media separation processes are a modern practical and economical adaptation of the well-known laboratory procedure for separating a mixture of two solids by immersing the mixture in a liquid having a specific gravity intermediate the specific gravities of two solids. The lighter solid floats while the heavier sinks. This method of separation has been attempted on a commercial scale, but the high loss and high cost of the organic liquids halted the development of the process.

Many attempts have been made to simulate a heavy liquid by using a suspension of a finely divided solid in water. If the solid phase of the suspension is ground fine enough, the suspension can be made stable or so slow settling that a substantially uniform specific gravity can be maintained from top to bottom of the bath. However, any material separated by such methods will inevitably be contaminated by some slime which will eventually accumulate in the bath and cause a viscous medium at the expense of separating efficiency. Therefore, it is necessary to provide means for continually cleaning a portion of the medium to eliminate slime at the same rate at which it is introduced to the medium. The problem of efficiently cleaning the medium limits the minimum grain size of the solid of the suspension in the case of the Chance sand process for cleaning coal, because decantation is the only cleaning method available. If the sand is too fine, it will be lost along with the slime. Therefore, coarse sand must be used, and to maintain a semblance of a uniform suspension, it is necessary to use strong rising water currents. The combination results in a separation based more on hindered settling classification than on sink-float principles.

As previously mentioned, galena was used as the solid constituent of the medium during the early stages of the development work. The high specific gravity of galena made it suitable for the preparation of medium for high specific gravity separations. Galena can be cleaned by either decantation or by froth flotation. As with sand, decantation limits the minimum particle size of the media that can be cleaned without excessive loss. Froth flotation for cleaning galena medium has been used, but the problem of floating fine galena that has been exposed to extensive oxidation is well known to be a most difficult one. Last year the largest heavy-media plant in the world, and the second plant to be installed, converted from galena medium to ferrous medium despite the fact that the ore contains galena which can be used as medium. The change to ferrous medium has been beneficial in many ways. Today all the heavy-media plants have been converted from galena to ferrous media.

Unquestionably, ferrous media have the widest application of any media developed, for the following reasons:

1. Ease of recovery and cleaning by magnetic means. Particle size or surface condition not a factor.
2. Low consumption per ton of ore treated.
3. Resistance to abrasion.
4. Widest range of media densities, including higher workable densities (1.25 to 3.4) than have been found possible with nonferrous media.
5. Space required for recovery and cleaning of ferrous media is considerably less than that for nonferrous media.
6. Ferrous media require lower capital investment and operating costs for media recovery and cleaning.

Advantages of Heavy-media Separation Processes

Heavy-media separation processes offer the following positive advantages, amply demonstrated on a wide variety

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of ores—nonmetallic as well as metallic, and coals.

1. Ability to make sharp separations at any predetermined specific gravity ranging from 1.25 to 3.4; and continuously maintain this preselected specific gravity within plus or minus 0.01. Efficient cleaning methods eliminate viscosity problems.

2. The specific gravity of the separating medium can be changed at any time, and within a few minutes, when necessary to meet changing characteristics of feed.

3. Ability to remove sink, or refuse, continuously.

4. Ability to treat full-size range material without presizing.

5. Heavy-media separation plants can be started and shut down without loss of values or operating efficiency.

6. The medium used for separation is relatively low cost and losses incident to the operation are negligible (0.2 to 0.8 lb per ton of feed). Since the medium can be recovered from the separated products almost completely, loss of values and medium due to the inefficient presizing caused by variable moisture in the feed is no longer a serious consideration.

7. Low operating and maintenance costs.

8. Separatory units have large capacity and occupy relatively small space.

9. Capital cost low in comparison with installation of competitive processes of equal capacity.

Scope of Heavy-media Separation Processes

GENERAL FIELD OF APPLICATION

In general, it may be stated that heavy-media separation processes will treat any ore in which the valuable mineral constituents have an appreciable difference in specific gravity from the worthless gangue. This difference in specific gravity can be less than that required for efficient separations by jigging or tabling.

In general three types of ore are amenable to heavy-media separation processes:

1. Ores whose valuable constituents are liberated from unwanted mineral by coarse crushing.

2. Ores containing valuable minerals in the form of lenses or veins separated from each other by barren gangue.

3. Ores containing barren gangue which can be rejected after coarse crushing.

The heavy-media separation processes are capable of performing three main functions.

1. Rejection of a waste product leaving an enriched product for further concentration by other methods after reduction in size.

2. Production of a finished concentrate and a rejectable waste in one operation.

3. Production of a finished concentrate, and a low-grade reject for additional treatment.

Heavy-media separation processes have a wide field of usefulness as preconcentration methods for removing gangue, at low cost, after coarse crushing. Therefore, they offer a cheap and effective means for making ore out of sub-ore, waste dumps and tailings, by concentrating the valuable constituents into a smaller product of sufficiently high grade to warrant treatment by more expensive methods such as flotation or cyanidation.

The application of heavy-media separation processes is related closely to the type of mining practiced or contemplated. It is, therefore, necessary to study the geological structure of the deposit to determine whether high cost selective mining methods may be eliminated through use of these processes to reject waste rock mined over greater widths. Similarly, by locating the heavy-media plant underground a hoisting bottle neck may be eliminated, at far less expenditure of time and money than would be required to sink and equip a new shaft. Installation underground, moreover, provides a waste product for back filling at no additional cost.

Of particular importance to established operations are the opportunities through heavy-media separation processes for increased profits by lowering of costs, for increases in production, and for reduction in overall labor requirements per ton of metal or mineral produced. An existing flotation mill, for instance, may greatly increase its metal output through increasing mill heads by installing heavy-media separation before fine grinding.

RANGE OF SIZES AMENABLE TO HEAVY-MEDIA SEPARATION

In general, it may be stated that heavy-media separation processes are applicable to the separation of sizes down to as small as 35 or 48 mesh. Although in some instances it is feasible to handle run-of-mine ore without presizing, the efficiency of separation drops off on sizes below 48 mesh.

For the treatment of sizes up to 3¼ in. a separatory cone may be used. Other apparatus is available for treatment of sizes larger than 3¼ in.

Regardless of the type of separatory vessel used, the underlying principles of heavy-media separation processes remain unchanged, although the flowsheet may vary. In this connection, as will be explained later, the treatment of full size range material, including sizes below 10 mesh, necessitates the use of more equipment than is required for handling sizes down to 10 mesh only.

RANGE OF SEPARATING GRAVITIES OBTAINABLE

Highly efficient separations have been made on coal at specific gravities as low as 1.25, using magnetite medium. With this medium it is possible to make highly effective separations up to 2.20 sp gr. For the range 2.20 to 2.90, mixtures of magnetite and ferrosilicon are satisfactory. Above 2.85, ferrosilicon alone is used and separations at 3.40 sp gr have been made commercially.

CHARACTERISTICS OF MAGNETIC MEDIA

Magnetite (Fe_4O_4) is a naturally occurring mineral found in abundance in many countries and is mined as an iron ore.

Ferrosilicon (Fe 85 pct, Si 15 pct) is a furnace product obtainable from various suppliers in several grades of suitable fineness.

Operations of Heavy-media Separation Processes

The essential steps in heavy-media separation processes are.

1. Preparation of the feed.

2. Heavy-media separation.

3. Removal of medium from the separated products.

4. Reclamation and cleaning of the medium for reuse.

PREPARATION OF FEED

Primary slimes, and fines which are not amenable to sink-and-float separation, are objectionable because they dilute the medium and increase its viscosity. Such fines and slimes should

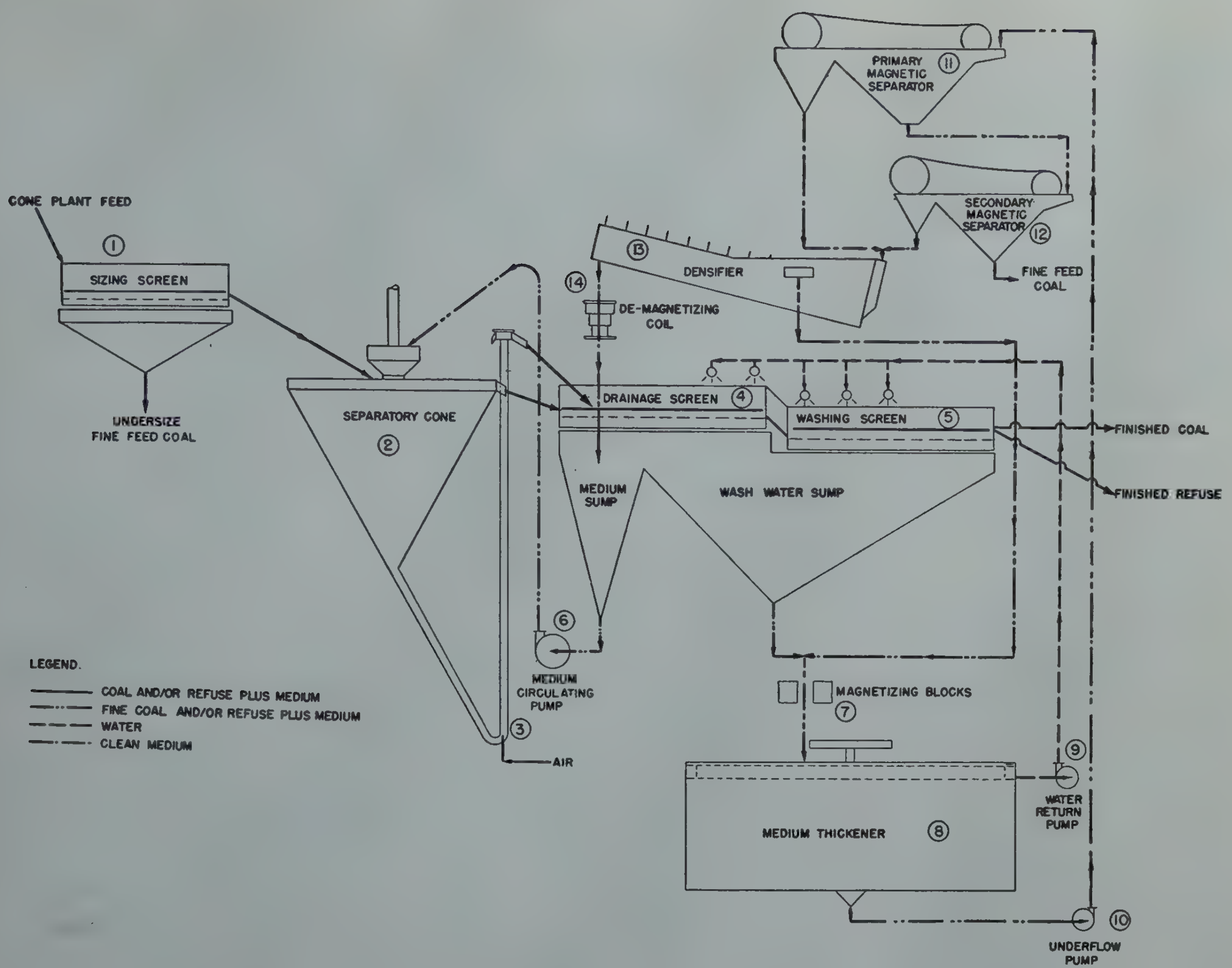

FIG 1—Standard flowsheet for sizes coarser than 10 mesh, using ferrous media.

be removed as completely as possible prior to heavy-media separation.

Fine granular or colloidal particles having a lower specific gravity than that of the solid constituent of the medium have two definite disadvantages:

1. They lower the specific gravity of the medium.

2. They reduce the separating efficiency of the medium because of increased viscosity.

The harmful effect of slime can be reduced to a great extent by the use of efficient methods for cleaning the medium. The development of ferrous media and magnetic recovery have eliminated the need for complete slime removal before treatment because of the efficiency of magnetic concentration. At one operation using ferrosilicon, the ore treated is high in slime content. The only equipment employed for 125 tons per hour is one 4 by 12 ft screen fitted with washing sprays for the first 6 ft only. This treatment only partly deslimes the ore, in fact a substantial quantity of clay balls is sent with the ore to the heavy-media cone. With other types of sink-float processes, the treatment of such slimey ore tends to result in medium contamination and high viscosity, but with ferrous media and magnetic cleaning, the slime is rejected from the medium as rapidly as it is introduced and a satisfactory separating medium is maintained without difficulty.

A clearer understanding of the basic principles of separation, and the method of recovery and cleaning of media can best be obtained by referring to the flowsheet shown in Fig 1. This flowsheet is the one most commonly used for treatment of sizes coarser than 10 mesh, using ferrous media.

For the treatment of sizes finer than 10 mesh, a modified flowsheet is used wherein separate medium cleaning circuits are employed for selectively separating and recovering medium from fine float and fine sink. This innovation is discussed later.

Referring to Fig 1, it will be noted that the finer sizes are removed from the feed by the primary washing screen (*1*) before it enters the separatory cone (*2*).

HEAVY-MEDIA SEPARATION

In the separatory cone (*2*) which is filled with ferrous medium of preselected gravity, the low gravity minerals float on the surface of the medium and are removed by overflowing a weir opposite the point of entry of feed. The heavier gravity minerals sinking through the medium are removed continuously from the cone by means of an "air lift" (*3*). The latter, as will be noted, is, roughly, a *J*-shaped pipe of suitable size connected to the bottom of the cone. Jets of compressed air introduced at the bottom of the vertical section of this air lift serve to elevate the sink particles in an effective and economical manner.

REMOVAL OF MEDIUM FROM SEPARATION PRODUCTS

The float and sink products discharged from the cone go to drainage

and washing screens, (*4*) and (*5*). In the particular flowsheet shown in Fig 1, the float and sink products are treated on single drainage and washing screens divided longitudinally by a partition to keep the products separate. In large size plants, separate drainage and washing screens are provided to handle the float and sink products.

The purpose of the drainage screen (*4*) is, as the name implies, to drain off and remove the medium from the float and sink products. Normally, more than 90 pct of the medium discharged from the cone with the float and sink products drains off on the screen or screens mentioned, and is returned directly to the cone, without further treatment, by means of one or more centrifugal pump (*6*).

The float and sink products pass along the drainage screen, or screens (*4*), to the washing screen, or screens (*5*), where substantially complete removal of the adhering medium is accomplished by means of water sprays as indicated in Fig 1.

The screen oversize products are discharged as concentrate or tailing.

Positive and simple control of the desired cleanliness and separating efficiency of the medium removed by the drainage screens and circulated directly back to the separatory cone is provided by a readily adjustable diverting chute underneath the drainage and washing screens (*4*) and (*5*). By means of this chute, more or less medium may be diverted from the drainage hopper to the washing hopper and thence to the cleaning circuit.

RECLAMATION AND CLEANING OF MEDIUM FOR RE-USE

The medium removed by washing is too dilute and, in many instances, too contaminated with fine impurities to permit of its being returned directly to the separatory vessel. Accordingly it is treated in a novel manner as follows.

The undersize from the washing screen (*5*) first flows by gravity, or is pumped, to a medium reclamation thickener (*8*). Just before it enters the thickener the diluted medium passes between a set of magnetizing blocks (*7*). These latter serve to change the charge on the discrete ferrosilicon or magnetite particles whereby they become mutually attracted and flocculate. The net result of this is faster settling of the particles in the thickener with consequent advantage of requiring less thickener area and depth than otherwise would be necessary. Permanent magnets are used for this purpose, but more recently electromagnets have been recommended because of the greater magnetic flux that they provide and because they are not subject to a gradual loss in strength due to heat, shock, and age as are permanent magnets.

The overflow from the thickener (*8*) may be used for spray water as shown in Fig 1. Alternatively, the overflow from this unit may be sent to another thickener which may serve as water-storage and clarification means for the entire operation. In the latter event, a portion of the overflow from this secondary thickener would be used for spray water over the previously mentioned washing screen (*5*).

In addition to serving as a means for dewatering the medium, the thickener (*8*) also provides storage for medium during off-operating periods. For this reason, this thickener is provided with a motorized device for raising and lowering the rakes. When the plant is shut down, the rakes are lifted to some point above the settled solids. When operations are resumed, the rakes are gradually lowered into the settled solids and then the latter are rendered sufficiently mobile so that they may be removed eventually by means of a centrifugal pump (*10*) and thereby sent to the cleaning circuit comprising at least two magnetic separators, (*11*) and (*12*).

When it is desired to increase the specific gravity of the medium in the separatory cone during operation in order to meet some change in characteristics of feed, an extra amount of medium can readily be introduced into the circuit by lowering the rakes in the thickener.

The magnetic cleaning circuit, as shown in Fig 1, comprises a primary (*11*) and a secondary (*12*) Crockett-type magnetic separator. These separators are of the belt type, the magnet assembly being mounted just above the lower section of the belt. Feed is added under the belt as shown and the fines, not immediately picked up by the action of the magnets and carried by the rubber belt, are dropped and eliminated through the hopper, as shown. This tailing, or reject, goes by gravity to the secondary magnetic separator (*12*) for additional treatment in order to recover any traces of medium not picked up in the primary separator. The tailing from the secondary separator (*12*) goes to waste; or if sufficiently valuable, may be sent to some other form of cleaning, such as froth flotation.

The medium recovered on the belt of the primary magnetic separator (*11*) is discharged into the hopper, as shown in Fig 1, after the belt passes the influence of the magnetic field. This recovered medium is too diluted with water to return directly to the separatory cone (*2*) and it is therefore treated in the following manner.

The primary magnetic separator concentrate flows by gravity, or it may be pumped, to a densifier (Fig 1, *13*). The densifier is essentially a screw-type classifier, but acts as a thickener and storage reservoir for clean medium. It is not a classifier in the accepted sense of the term. The machine must be fed at the end by means of a feed box, and overflows at the sides—just the opposite of a classifier. The screw is a double helix flight and is equipped with a variable speed drive.

Either a manually operated or motorized lifting device is provided to raise or lower the screw as required.

The densifier performs the important function of assisting in the regulation of the specific gravity of the medium in the separatory cone in addition to dewatering the medium recovered by the magnetic separators.

The medium removed by the secondary magnetic separator (*12*) may join the flow of medium recovered by the primary separator (*11*). Alternatively, it may join the overflow of the densifier (*13*) which returns to the thickener (*8*).

The screw discharge of the densifier returns by gravity to the separatory cone (*2*) via the drainage hopper and medium return pump (*6*). Before it reaches the drainage hopper, however, it passes through an alternating current demagnetizing coil (*14*). This is an essential step in the heavy-media separation flowsheet. The dispersing effect imparted to the particles of medium by the action of the demagnetizing coils gives fluidity to the medium in the separatory cone without which satisfactory operation could not be maintained.

As previously stated, the densifier (*13*) provides a means for ready adjustment of the specific gravity of the medium in the separatory cone. The medium discharged by the screw is of considerably higher specific gravity than the medium in the cone. Therefore, water may be added to the medium returned to the cone by the pump (*6*). Ordinarily, any required

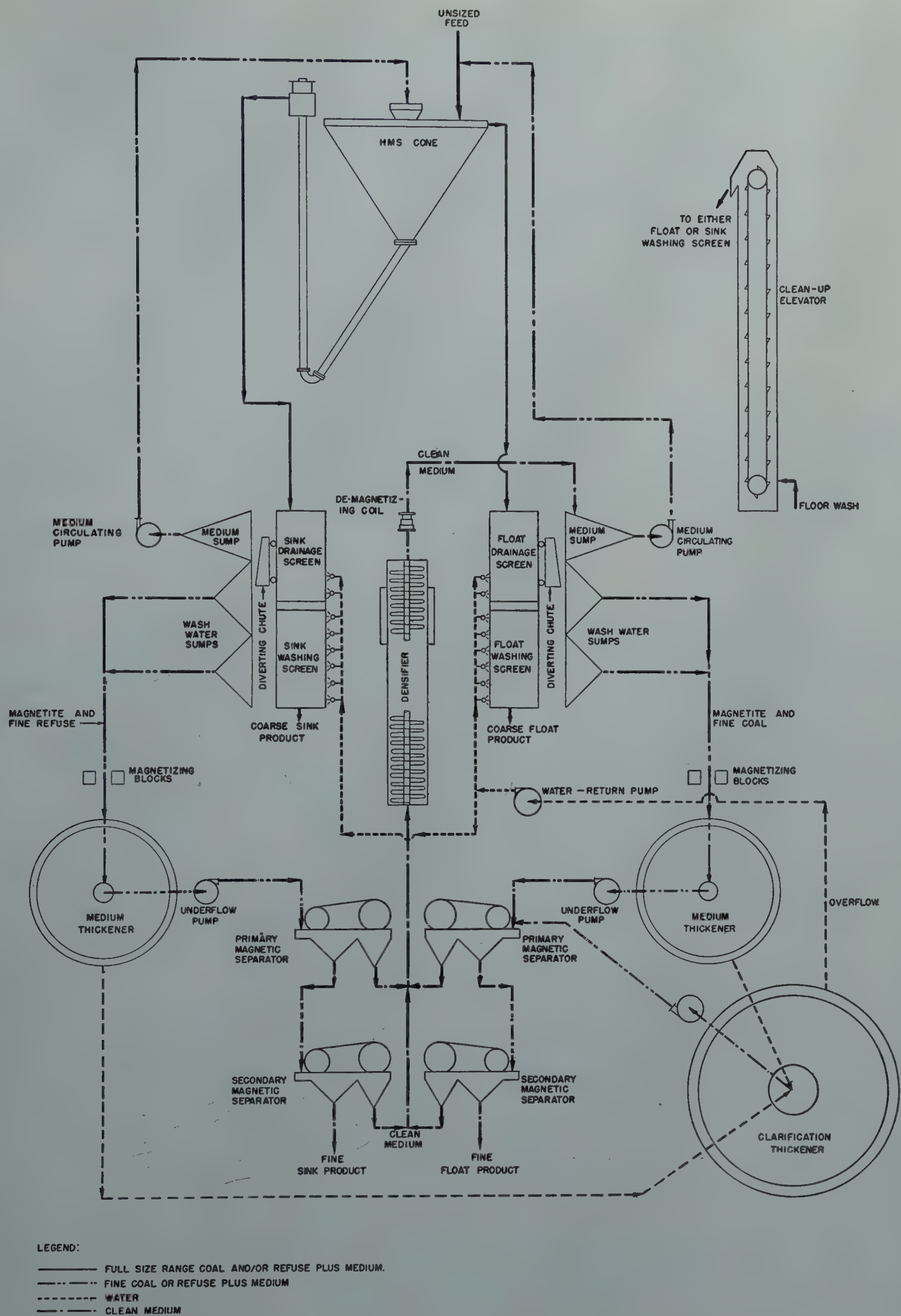

FIG 2—Flowsheet of treatment commonly used for sizes below 10 mesh.

adjustment of specific gravity of medium in the cone is most quickly and easily done by simply turning a valve to regulate the flow of water.

A bucket elevator, not shown in Fig 1, is used to introduce new medium into the circuit, and is also used as a means of returning any clean-up material from floor washing to the system. When necessary to dump the separatory unit, this elevator is used to return the medium, as well as the intermixed ore, to the circuit. In some plants a pump is used to advantage instead of an elevator.

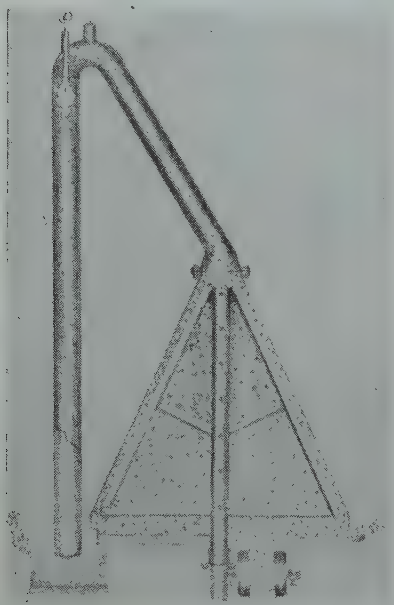

FIG 3—Separatory cone used in heavy-media separation.

Application of Heavy-media Separation to Fine Sizes

The results of recent continuous pilot plant testing at Stamford have shown the possibility of extending the application of heavy-media separation to the treatment of material as fine as 28 to 48 mesh.

For the treatment of sizes below 10 mesh, a modified flow scheme has been developed. Essentially, this involves the use of dual magnetic cleaning circuits wherein the undersize float and the undersize sink products from the screens are separately subjected to magnetic separation. The magnetic separators recover the medium for re-use and the tailings from these separations are either a finished float or a finished sink product, as the case may be.

The flowsheet shown in Fig 2, describes the equipment used and the flow of products and medium through the circuit.

It will be noted that just as in the case of the flowsheet shown in Fig 1, a certain portion of the medium removed on the drainage screens is returned directly to the separatory cone. The amount thus returned is controlled by

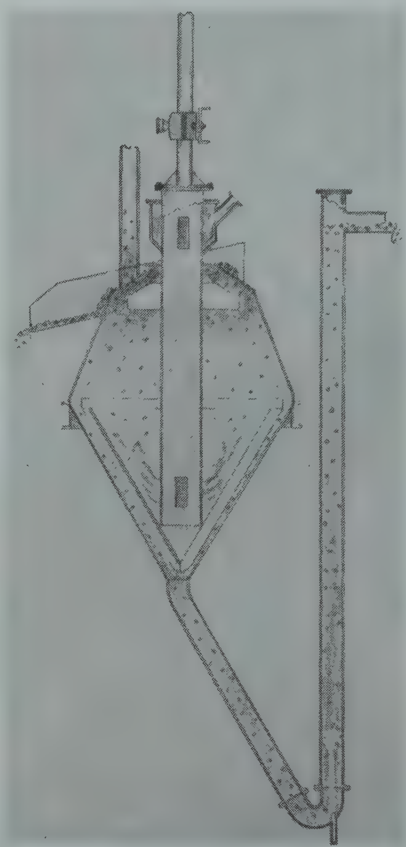

FIG 4—Closed-top cone used in heavy-media separation.

the position of the diverting chute and is dependent upon the quantity of undersize float and sink in the original feed. It will be apparent that when treating a feed containing a large proportion of undersize material, it becomes necessary to divert a greater amount of drainage medium to the cleaning circuit than would be the case when the cone feed contains a smaller proportion of fines. In other words, the amount of fine material that can be recirculated to the cone with the medium depends on just how much can be tolerated in the cone without interfering with the separating efficiency of the medium. Eventually, of course, all of the fine material in the separatory cone circuit reports as finished float or finished sink. The advantage of recirculating a portion of the fines with the drainage medium is that a saving in the size of the magnetic separator units is thereby effected. Instead of having to clean at one time all of the medium and accompanying fines discharged from the cone with the float and sink products, only a portion of it goes to the magnetic separators; the remainder is recirculated to the cone.

Separatory Vessels

The heavy-media separatory cone previously shown in Fig 1 is illustrated in more detail in Fig 3. This open top cone has been found most suitable for separations where a large amount of float has to be removed. In other cases,

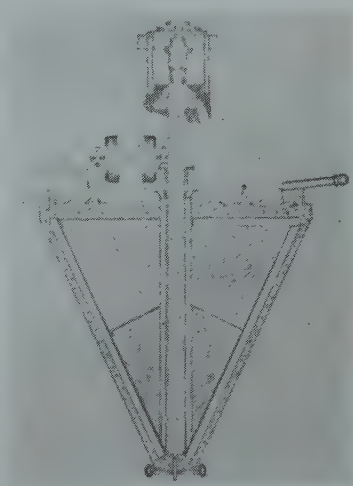

FIG 5—Inside air lift cone used in heavy-media separation.

particularly for the treatment of iron ores where a finished sink product is required and where the amount of float to be removed is relatively smaller in proportion to the sink, a closed top cone as shown in Fig 4 is used. A third type of cone, shown in Fig 5, employs an inside air lift.

All three types of cones are in operation in heavy-media separation plants.

The cone separator fitted with an inside or an outside air lift is an efficient, flexible, and economical vessel.

The scope of the heavy-media separation processes is not limited to the use of cone-shaped separators, however. Other shapes have been developed and used for ores presenting special problems of treatment.

The Colorado Iron Works have adapted the Akins classifier for use as a

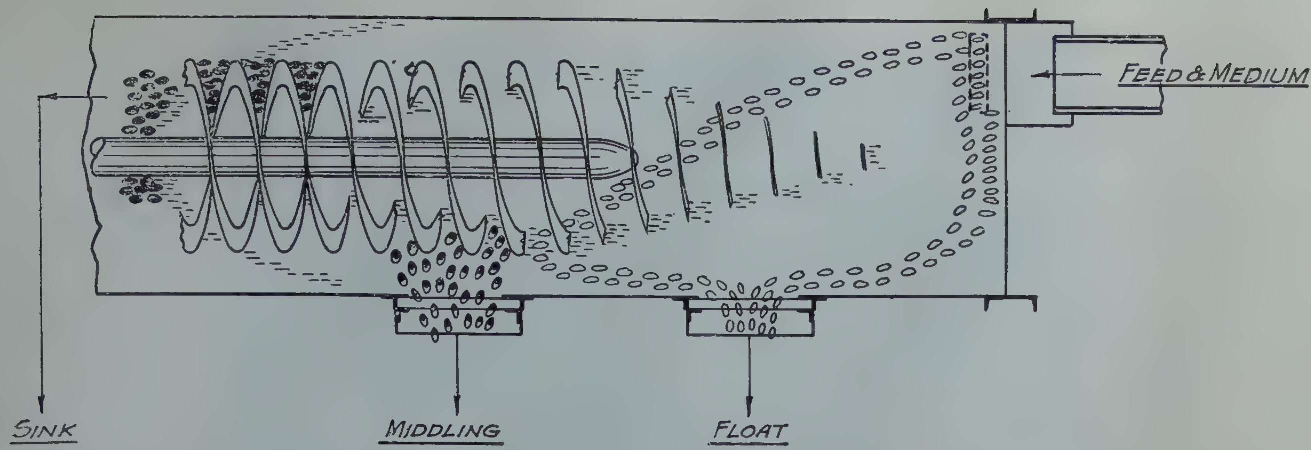

FIG 6—Akins classifier adapted for heavy-media separatory vessel.

heavy-media separatory vessel. This unit is illustrated in Fig 6 and has had outstanding success treating ores of the iron range in Minnesota. The principal advantages of the vessel are reduced head room and elimination of the air lift for sink removal. The plant can be shut down at the end of the shift, or in event of power failure, without draining the vessel.

A new separatory vessel has been developed by the engineers of the Link-Belt Co. This machine is illustrated in Fig 7. It was designed particularly for the treatment of large size coal. A unit

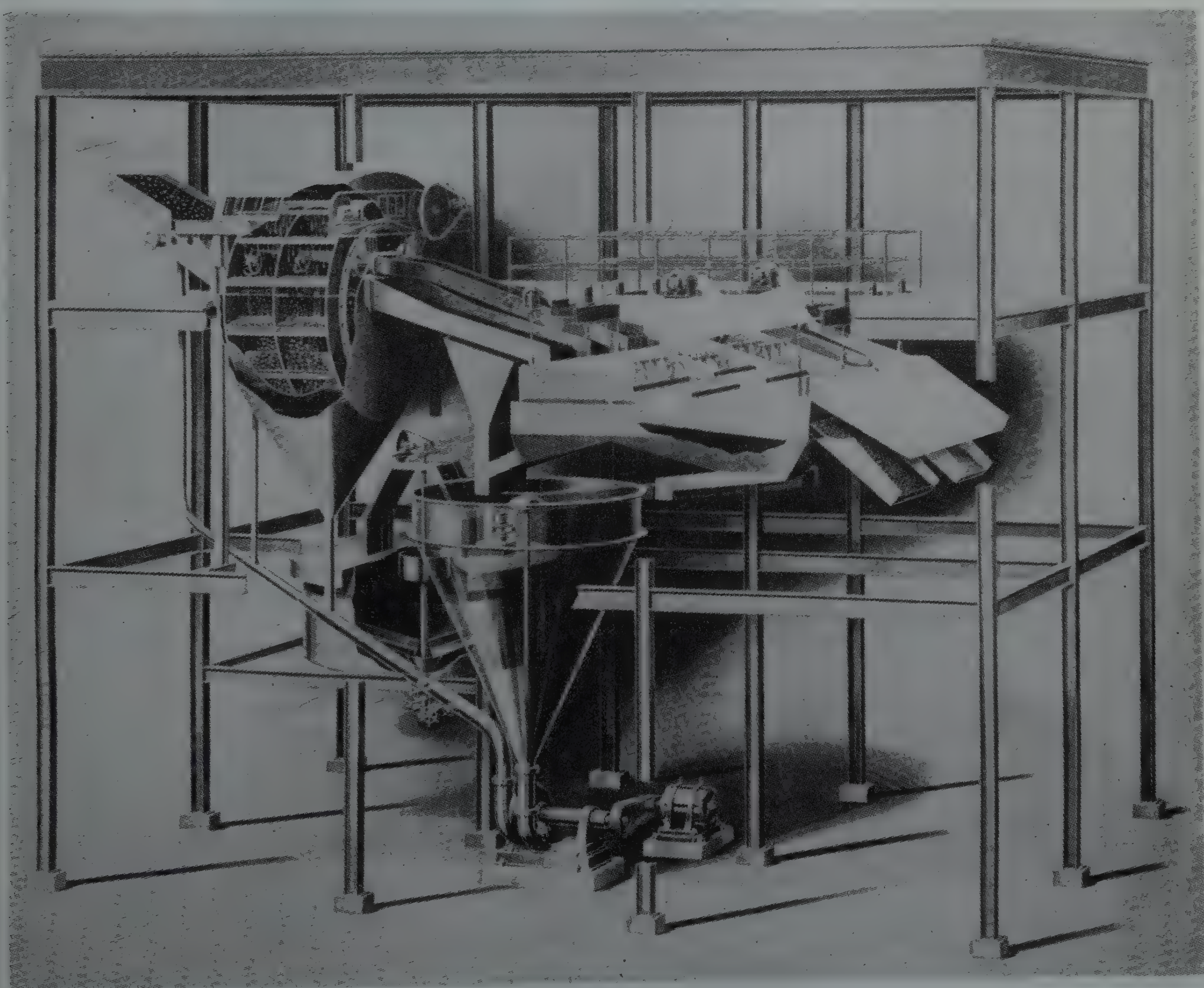

FIG 7—Link-Belt float-sink concentrator showing separatory vessel in upper left.

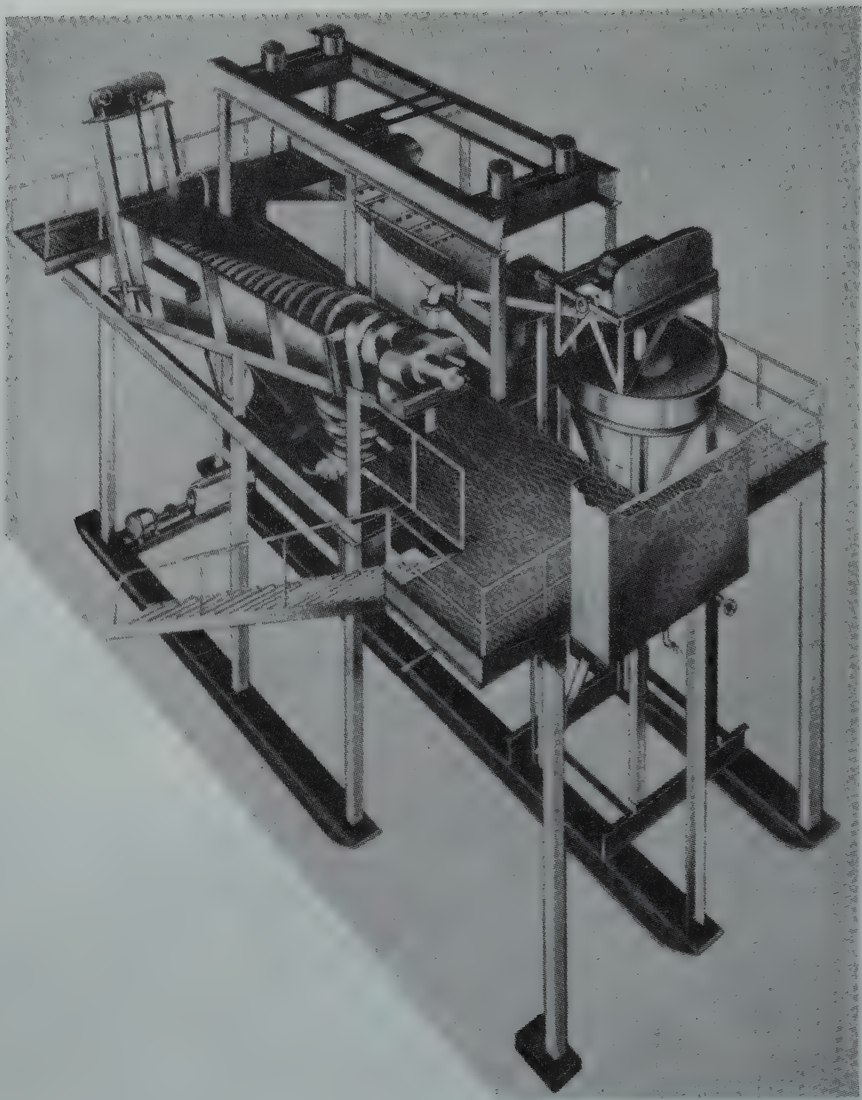

FIG 8a—Typical portable heavy-media separation unit.

was installed in an operating coal cleaning plant in the Pittsburgh district about two years ago. The vessel has demonstrated its ability to handle mechanically large coal and large tonnages. Plans are now being made to install Link-Belt vessels in what will be one of this country's largest coal cleaning plants. Several smaller installations also are contracted for.

Small Tonnage Unit

Although heavy-media separation processes have found their widest application in operations treating substantial daily tonnages, they are equally applicable to small tonnage operations.

For concentrating ores from deposits with limited reserves, mine dumps, and retreatment of coarse gravity tailings piles, or to serve as a pilot plant for demonstrating the efficiency of heavy-media separation on current production or during development of new properties, a semiportable heavy-media separation unit has been developed. In this unit the essential pieces of equipment have been reduced to meet minimum requirements consistent with high separating efficiency in order that capital costs may be reduced to a minimum. As might be expected, operating costs per ton for the semiportable unit will be somewhat higher than for standard units treating tonnages at a higher rate. A typical unit of this type is the Mobil-Mill offered on a rental or sale basis by Western-Knapp Engineering Co., Division of Western Machinery Co., as illustrated in Fig 8. These small plants have found wide application in the industry. Six have been installed as production units at small mines and others are being used as pilot plant installations at producing shafts. Rock from various sections of the deposit can be tested as well as mine output resulting from various mining methods. In this way the most advantageous mining method, economic limit of ore grade, and value of preconcentration by heavy-media separation can be determined and can be coordinated for the most desirable overall operation.

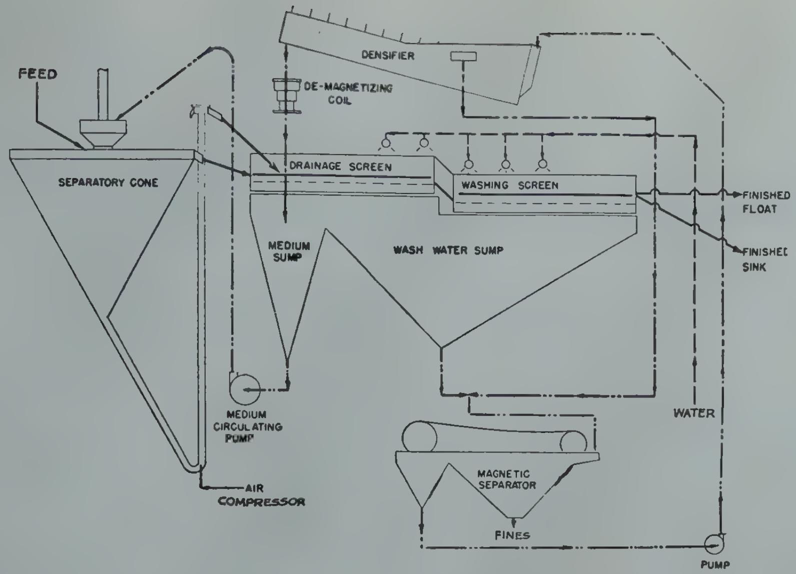

FIG 8b—Diagram of portable heavy-media separation unit.

Separatory Medium

The ideal separating medium obviously would be one having the low viscosity and high stability (uniform gravity from top to bottom) of heavy liquids. When dealing with a suspension of a solid in water both these ideals cannot be attained. Reduction of the particle size of the suspended solid will decrease the settling rate of the solid and thus increase the stability of the suspension but will increase the viscosity because of the increased surface area generated. A compromise must be reached, therefore, between stability and viscosity. The open-top cone shaped separator permits sacrifice of stability so as to maintain the low viscosity essential for maximum settling rate of fine heavy ore particles and hence high tonnage and accurate separation. The unstable suspension results in a differential density; the specific gravity at the bottom of the bath will be higher than at the top. The differential between top and bottom gravities will normally be in the neighborhood of 0.10 to 0.25 in the case of a 20 ft diam cone, which with 60° sides would be over a vertical distance of 17 ft. Obviously the differential existing in top 6 or 8 in. of the bath where the actual separation takes place will be exceedingly small and cannot conceivably influence the separation. In any vessel other than a cone, such a differential density could not be tolerated because any ore would contain some particles having a specific gravity between the top and bottom gravities in the cone, and such pieces would come to rest or teeter when they reached a zone having a specific gravity equal to their own. An accumulation of such material would ultimately plug the cone if it were not for the differential downdraft created by the operation of the air lift. Air lifts normally are designed to have a pipe velocity of about 6 ft per second. This velocity exists at the intake opening of the air lift. At various horizons above the air lift intake, the downdraft diminishes in proportion to the diameter of the cone at such horizons. The medium circulated by the air lift is returned to the cone at any desired level or levels. In this way the downdraft of the air lift can be satisfied before its influence is felt in the top 6 or 8 in. of the bath which is the separating zone.

From the foregoing, it will be seen that the combination of ferrous media, magnetic cleaning and control, and the cone with air lift combination has eliminated the difficulties with which the various sink-float methods have been confronted.

Magnetic cleaning permits the use of the size and type of medium best suited for any operating gravity between 1.25 and 3.40 so as to develop a reasonably stable medium.

Operating Industrial Mineral Plants

Many successful applications of heavy-media separation to industrial mineral beneficiation problems have already been established.

At Chewelah, Wash., the Northwest Magnesite Co. is beneficiating approximately 150 tons per hour of magnesite mixed with dolomite, shale, and quartz. The quarry product is crushed to minus 1½ in. and the minus 3/16-in. portion is removed. The minus 1½ in. plus 3/16 in. rock is fed to a 20 ft diam outside air lift cone. The specific gravity of the separation medium is varied between 2.80 and 3.00 depending on the grade of magnesite desired.

At this operation heavy-media has replaced a large group of hand pickers who formerly picked magnesite from broken rock at the quarry face. The economy of operation and the efficiency of separation obtained by heavy-media has made it possible to retreat profitably the rejects from the hand picking operation.

At Rosiclare, Ill., the Rosiclare Lead and Fluorspar Mining Co. is treating in the neighborhood of 40 tons per hour of fluorspar ore. The flexibility of

heavy-media provides an accuracy of control over the product grade that was never obtained with the jigs previously used. In addition, a substantial tonnage of metallurgical-grade fluorspar has been recovered by retreatment of old jig tailings. Operating costs have been reduced, recovery increased, and accurate product control made possible.

Most of the other fluorspar producers in the Rosiclare district have adopted heavy-media.

An unusual industrial mineral beneficiation problem is the recovery of garnet from hornblende at the Barton Mines Corporation property at North Creek, N. Y. Approximately 40 tons per hour of minus 1¼ in. plus ⅛ in. quarry product is treated at a specific gravity of 3.17. Heavy-media replaced jigs at this operation. Costs have been reduced and recovery improved.

The application of heavy-media separation to coal cleaning problems is now being studied. A pilot plant with a capacity of 250 tons per hour has been in operation in one of the Pittsburgh Coal Company's plants for over two years. The feed to heavy-media is 4 by ⅜ in. secondary middling from an older coal cleaning process. The ability of heavy-media to recover clean coal from such a product has demonstrated the efficiency of the process. Several large tonnage heavy-media plants are now under construction or are being designed for coal treatment, both bituminous and anthracite.

Laboratory tests followed by a year's operation of a 100 ton per hour pilot plant have demonstrated the superiority of heavy-media over older methods for the recovery of diamonds in Africa. A 12,000 ton plant has been designed and is now under construction.

A barite producer in Southeastern United States had actually been forced to suspend operations because of the inability of jigs to produce a barite concentrate meeting market specifications. The installation of a heavy-media separation plant allowed resumption of operations. Market specifications are now met without difficulty even when treating refractory material previously by-passed.

Although heavy-media separates fragmentary materials on the basis of specific gravity and not of particle shape, sometimes we find mixtures where one rock type will break to an undesirable shape and also have a specific gravity differing from the rest of the mass. This has interested several producers of concrete aggregate who are troubled by excessive "flats." Laboratory tests have demonstrated the ability of heavy-media to eliminate "flats" from aggregate because in most cases a substantial difference in specific gravity exists.

A heavy-media separation plant is now in satisfactory operation at the site of a large concrete project. Local gravel is being cleaned of undesirable rock types, and concrete made with this aggregate has been pronounced excellent after government laboratory tests. The elimination of an expensive freight haul has made this a most profitable operation.

Laboratory Testing

At the Research Laboratories of the American Cyanamid Co., at Stamford, Conn., the Mineral Dressing Division maintains equipment capable of treating from 200 lb to as high as 5 tons per hour by heavy-media to determine the amenability of any type of material to beneficiation.

A wide variety of materials have been found amenable to treatment and a number of new plants are being designed, or are under construction.

Gypsum ores from England, Canada, and the United States have been successfully treated. Of particular promise in the treatment of gypsum was the development of accurate product control through specific gravity control. Heavy-media promises a gypsum producer the ability to vary the quality of his product so as to meet market demands.

Bauxite ores have responded to treatment in some cases. Both high-iron and high-silica types have been substantially beneficiated.

A spodumene producer in Western United States has had small scale heavy-media tests conducted on his ores. The results obtained have been so satisfactory that a plant is now being installed and will be in operation in the near future.

Tests on a Canadian corundum ore have shown the ability of heavy-media to produce a satisfactory product from low-grade rock.

Potash ores have been treated by heavy-media using a suspension of magnetite in saturated brine and three product separations have been made in several cases—a coarse finished product, a coarse reject, and a middling product requiring fine grinding prior to froth flotation or leaching.

One producer of railroad ballast has shown considerable interest in heavy-media separation because laboratory tests have demonstrated the ability of the process to eliminate a low specific gravity rock from his quarry output. The use of full face mining followed by heavy-media offers cost reduction, accurate product control, and greatly extended quarry life to this producer.

A I M E

Texas White Firing Bentonite

BY FORREST K. PENCE

Location, General Geology, and Development

BENTONITE deposits are known to occur in Texas within the Jackson group of formations. This group represents the uppermost Eocene age sediments found in the coastal plain area of Texas. It outcrops across this area of the state in a narrow band of some 4 to 20 miles width. The outcrop pattern roughly parallels the present Gulf of Mexico shore line and is some 100 miles inland from the Texas shore, Fig 1.

The principal bentonite deposits are found in the areas where this outcrop pattern cuts across the south-central Texas counties of Karnes, Gonzales, and Fayette. In these deposits, the better quality bentonite is found in the lower or bottom layers of the volcanic ash deposits in which they occur.

Some of these better quality bentonites develop very light colors upon firing and therefore justify their being classified as "white firing." The deposits in Karnes and Gonzales Counties apparently occur in commercial quantity, whereas the white firing strata so far uncovered in Fayette County have been too thin to be classified as yet as "commercial." A study of the ceramic properties of the weathered ash in Gonzales and Karnes Counties was reported in 1941.[1]

Commercial development of the deposit in Gonzales County, 7 miles east of Gonzales, Texas, was started earlier by the Max B. Miller Co. for the purpose of marketing the material as a bleaching clay, and this operation has developed to very sizable proportions. In recent years, this company has offered a specially selected grade of the Gonzales material as a suspending agent in glaze slips. The white firing property especially adapts the material to use in white cover coat enamels.

The strata in the deposit are practically horizontal and consist from top to bottom of approximately 2 ft of soil overburden, 10 ft of brown bentonite, 2 ft of coarse white bentonite, and 4 ft of waxy white bentonite overlying a fine grained sandstone. The cut being made in the quarry is approximately one-half mile in length. Only the bottom 4 ft of waxy bentonite is being recovered, the upper layers being stripped and wasted, Fig 2. It may appear somewhat surprising that the very bottom strata appears to have been the one most completely altered.

To confirm this, samples from top to bottom of the various strata were studied microscopically by R. F. Shurtz, Professor of Ceramic Engineering, University of Texas. His interpretation is to the effect that the lower part of the seam was deposited at a much earlier date than the top, and that the lower part was chemically altered to a considerable extent before the upper part of the seam was laid down. The conclusion to be derived from these examinations may be stated briefly to be that the alteration in these strata or parts of strata has proceeded independently of the alteration in other parts of the strata during a considerable geological period. The presence of gypsum and iron stain throughout all of the strata indicates that alteration is now proceeding more or less uniformly throughout. It is contended that the alteration of the original ash to montmorillonite is not a result of the presently operating processes.

Columbus Meeting, October 1946.
TP 2504 H. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before March 30, 1949. Manuscript received Nov. 4, 1946; revision received Aug. 24, 1948.

* Department of Ceramic Engineering, University of Texas, Austin, Texas.

[1] Forrest K. Pence: White Firing Texas Volcanic Ash as a Body Ingredient. Amer. Ceramic Soc. *Bull.* (1941) **20** (10) 327.

A deposit which occurs approximately 7 miles southeast of Falls City and just south of the village of Castahowa, has been explored and leased by J. R. Martin, of San Antonio. Mr. Martin has conducted mining and marketing operations in bentonite for a period of many years and asserts that the white firing strata found in this deposit occurs in commercial quantities. His pit, which is shown in Fig 3, exposes 2 ft of soil overburden, approximately 5 ft of white bentonite having coarse texture, and approximately 5 ft of waxy white bentonite which in turn overlies a brown sandy clay. Here, as in the Gonzales deposit, the most completely altered portion is found at the bottom of the seam, as per following report of microscopic examination by Mr. Shurtz.

Sample No. 1: This sample was taken from the top stratum which is one foot thick. It is grayish in color and it contains visible fossilized plants. The color is probably the result of fine carbonaceous material in the rock. Under the microscope the sample is seen to consist of glass and feldspar; the amount of glass predominating. Both these substances are slightly altered. No montmorillonite or other clay mineral can be identified definitely; however, the products of the slight alteration mentioned are probably montmorillonite or mineral gel.

Sample No. 2: This sample was taken from the stratum second from the top. This stratum is fourteen inches thick. The sample is light gray. It shows numerous veinlets of greenish translucent material ranging from one-eighth inches wide down to the limit of visibility with the unaided eye. It has the smooth, sub-conchoidal fracture characteristic of some bentonites.

Microscopically the sample consists mainly of aggregates of clay minerals. The birefringence of the aggregates is lower than would be expected if the

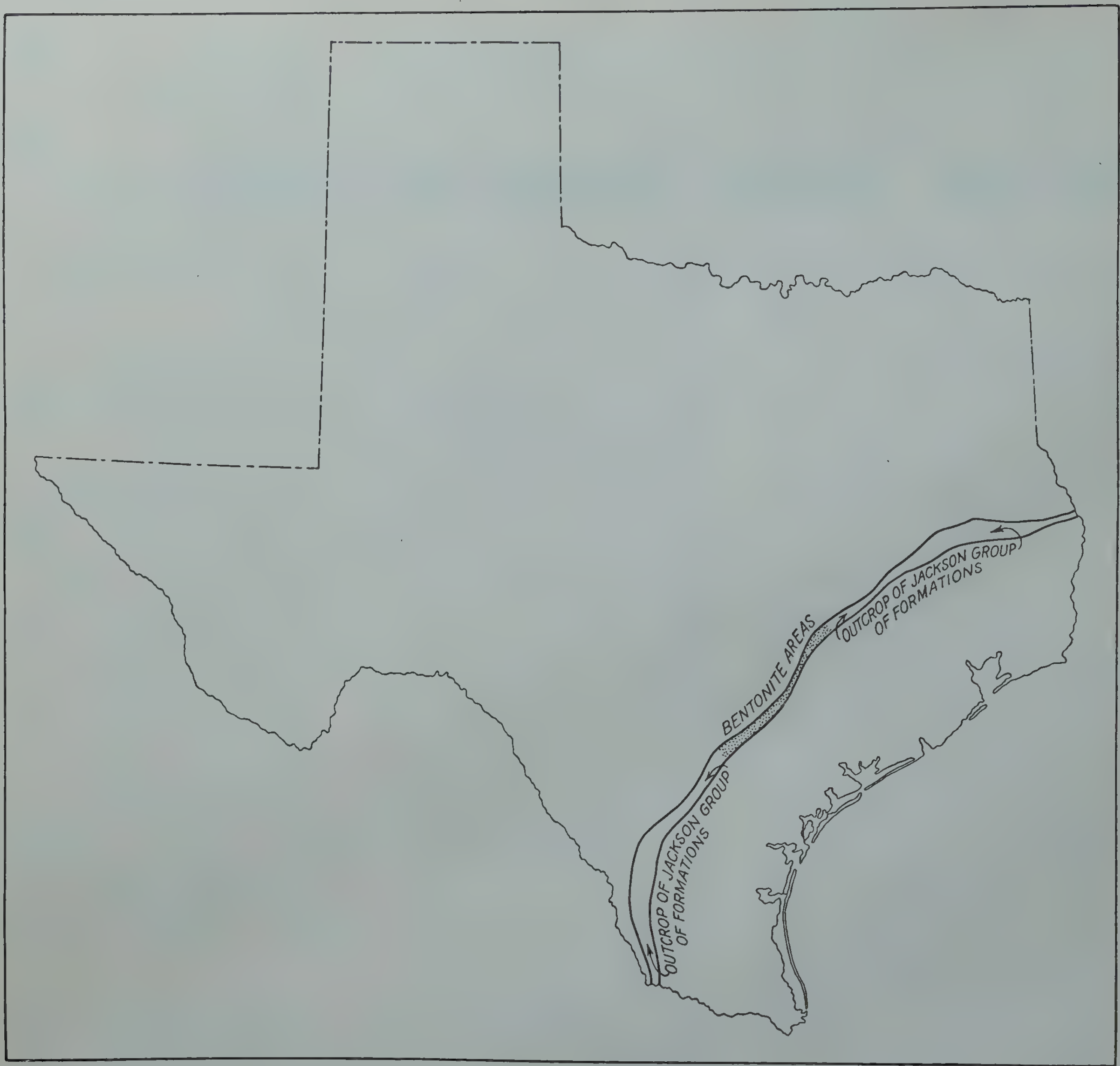

FIG 1—Map of Texas showing outcrop of Jackson group of formations and bentonite areas.

aggregates were pure montmorillonite; on the other hand, the birefringence is higher than expected if the aggregates were pure minerals of the kaolin family. The veinlets mentioned above furnish a clue for the explanation of the anomalous birefringence. Powder taken from the veinlets is isotropic, or nearly so, and the index of refraction is 1.48. These optical properties and the occurrence identify the material in the veinlets as a mineral gel similar to allophane. Inasmuch as the aggregates are not optically homogeneous, they are assumed to be intergrowths of montmorillonite and mineral gel.

Residual glass and feldspar are present in this sample in small amount.

Sample No. 3: This sample was taken from the stratum third from the top. This stratum is one foot thick.

The microscope shows some aggregates of clay minerals that are not noticeably different from those aggregates observed in Sample No. 2. However, there is much slightly altered glass and feldspar in this sample and the aggregates are in lesser amount than the glass and feldspar.

Sample No. 4: This sample was taken from the stratum fourth from the top of the deposit. This stratum is eighteen inches thick.

The microscope shows aggregates of clay minerals; but these aggregates have higher birefringence than those in Samples No. 2 and No. 3. For this reason the amount of montmorillonite is thought to be much greater than the amount of mineral gel. There are traces of glass and feldspar in the sample.

Sample No. 5: This sample was taken from the stratum fifth from the top of the deposit. The stratum is eighteen inches thick and is lowermost in the deposit.

The microscope shows this sample to be similar to Sample No. 4 in all respects except that the amount of glass and feldspar appears to be still less than the amount in Sample No. 4.

Summary: On the assumption that the volcanic ash in each of these five strata was originally more or less the same, the existing differences among the strata can be explained only by differences in the amounts and kinds of alteration to which the strata have been exposed. The three lower strata show alteration paralleling the order of deposition; that is, the lowest and oldest stratum is most altered whereas the uppermost and youngest of the three is least altered. This is quite understandable because, if alteration began immediately after deposition, the oldest stratum would be most altered. However, the second stratum from the top, younger than any of the lower three, shows alteration advanced much beyond that of the stratum below it. This fact cannot be explained by age alone and appeal must be made either to a change in the character of the ash or to a change in the altering environment or to both possibilities. The examination made and reported here was not sufficiently detailed to discriminate between these possibilities.

FIG 2—Exposure of Gonzales bentonite strata.

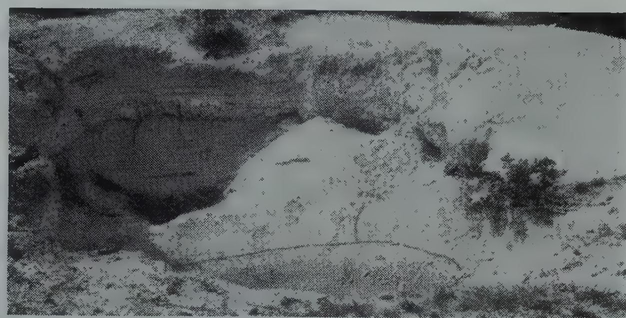

FIG 3—Exposure of Falls City strata.

Physical and Chemical Properties

All laboratory determinations reported in this paper were made on the lowermost strata consisting of waxy white bentonite. These strata also developed the whitest color in the firing tests. In the course of the investigation of the material in the ceramic laboratory, absorptions and shrinkages were determined over a range of temperature between Cone 07 and Cone 13. This study disclosed the fact that the bentonite at Gonzales is much more refractory than that at Falls City, as is shown in Table 1.

Table 1 . . . Fired Properties of the Raw Bentonites

Cone	Gonzales		Falls City	
	Fired Shrinkage, Per Cent	Absorption, Per Cent	Fired Shrinkage, Per Cent	Absorption, Per Cent
07	7.00	20.48	14.57	5.20
06	7.17	15.61	14.97	2.69
03	7.51	17.93	18.11	0.60
01	8.19	16.17	15.36	0.17
1	7.51	17.07	14.97	0.35
3	7.51	17.96	14.97	0.23
4	7.51	18.63	14.57	0.38
5	8.19	16.42	12.20	0.00
8	9.22	13.88		
9	9.39	13.64		
11	9.39	15.14		
13	11.43	10.02		

The study of the bentonites from the two quarries brought out in addition physical properties shown in Table 2.

Table 2 . . . Physical Properties of Bentonite from Gonzales and Falls City Quarries

Pyrometric Cone Equivalent (PCE): The method of testing followed the procedure specified by the A.S.T.M. designation: C 24–46. The PCE determination is made, therefore, by comparison of temperature of deformation of test cones made of the bentonites with corresponding deformation of standard pyrometric cones.

PCE

Gonzales bentonite...... Cone 15
Falls City bentonite..... Cone 9

Volume Changes in Swelling Tests: Comparisons were made of the swelling properties of the bentonites in question along with standard Black Hills bentonite and a typical domestic kaolin.

Material	Volume Increase, Per Cent	Volume Decrease, Per Cent
Black Hills bentonite...	700	
Gonzales bentonite	100	
Falls City bentonite....	40	
North Carolina kaolin..		30

The volume increase for the Gonzales bentonite is typical of those Texas bentonites which possess the swelling property. The Falls City would be classified as practically non-swelling.

Determination of Drying Shrinkage. Bars made up in the plastic form, determination made in the usual manner.

Material	Drying Shrinkage, Per Cent
Gonzales bentonite..........	5.9
Falls City bentonite.........	12.1

Chemical analyses were then made with the results given in Table 3.

Table 3 . . . Chemical Analysis of White Firing Bentonites

Component	Gonzales, Per Cent	Falls City, Per Cent
Ignition loss...........	9.16	12.70
Silica..................	71.00	66.64
Alumina...............	13.00	14.05
Ferric oxide............	0.85	1.19
Calcium oxide..........	1.20	0.46
Magnesium oxide.......	2.04	2.26
Sodium oxide..........	2.30	2.56
Potassium oxide........	0.60	0.60
	100.14	100.46

Examination of chemical analyses does not reveal sufficient differences in composition to explain the differences of the physical properties of the two bentonites.

Materials were then examined by thermal analysis, Fig 4. That from Falls City shows a large endothermic reaction at 1500°C and a small endothermic reaction at 700°C, which are characteristic features of the mont-

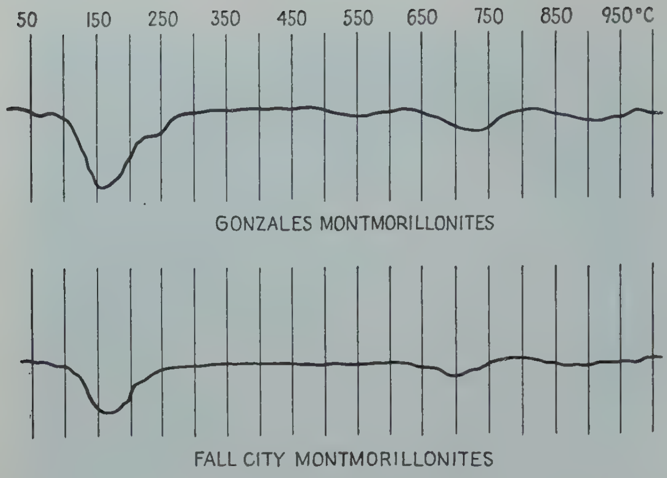

FIG 4—Thermal analysis curves of white firing montmorillonites (Texas).

morillonite series of minerals. That from Gonzales shows in addition to the two endothermic reactions (at 1500°C and 700°C) an endothermic reaction at 600°C and an exothermic reaction at 950°C, characteristic of a kaolin mineral.

Mineralogical Identification

A study of the two bentonites under the electron microscope indicates that the Falls City material consists of practically pure montmorillonite, whereas the Gonzales carries with the montmorillonite, halloysite as an accessory mineral. The oblong-like crystals, (Fig 6 and 7) identify the accessory mineral as halloysite. The nebulous particles seen in the micrographs are characteristic of montmorillonite.

As a final determination of the mineral composition, X ray diffraction patterns were made using the Norelco Geiger X Ray Spectrometer (Fig 5). The pattern for the Gonzales sample shows montmorillonite, halloysite and possibly crystobalite, as indicated by several lines that could be attributed to that mineral. The presence of crystobalite tends to be confirmed also by the high silica content (Table 3).

The pattern for the sample from Falls City checks closely with montmorillonite from Arizona which is an approximate standard of purity. There are a few lines in this pattern also that check with the data for crystobalite. The Falls City bentonite is, therefore, thought to be a montmorillonite with a moderate quantity of crystobalite.

Industrial Uses, Present and Potential, as Shown by Laboratory Tests

The Gonzales bentonite is being mined extensively and processed for use as a bleaching clay. It is thought that the Falls City material can be similarly utilized although the occurrence of Falls City is probably not as extensive as the one at Gonzales. Especially selected material from the Falls City deposit could also be used as a suspending agent in enamel slips, as is already being done with the Gonzales material as previously pointed out.

The white firing quality of the bentonites made them the object of special interest as ingredients in porcelain and semiporcelain compositions. In this investigation standard porcelain body compositions were made in which substantial quantities of the Gonzales bentonite and the Falls City bentonite were incorporated, with the result that it was shown that these materials gave a longer plasticity range and greatly increased drying strength and improvement in glaze fit. The body made with Gonzales bentonite has a high degree of whiteness, whereas, that using bentonite from Falls City tends to approach the ivory shade. Because of the strength and dense burning quality of the bentonite from Falls City, however, it could be used judiciously to replace ball clay and feldspar where a maximum degree of whiteness is not required.

These bentonites possess another quality which has yet to be overcome. They tend to give a sticky type of plasticity rather than the smooth, oily feel of normal ball clay. Further, in casting process the slip made of the powdered material and water tends to form gels that slow down the casting properties and cause sticking to the mold. This property of a slip is desig-

Table 4 . . . Body Compositions

	Test Number		
	EB-8	EB-9	EB-10
Troup ball clay[a]	25	25	25
Raw Gonzales bentonite	15	8	8
Davis Mountain kaolin	45 raw	52 raw	52 heated to 350°
Falls City bentonite (calculated at cone 06)	15	15	15
Dolomite	4	4	4
Determined data			
Total shrinkage	20.0	20.0	14.5
Absorption	6.5	4.6	4.8

[a] From Troup.

Table 5 . . . Body Compositions

	Test Numbers			
	EB-16	EB-17	EB-20	EB-21
Troup ball clay	25	25	25	25
Gonzales bentonite (raw)				5
Falls City (calulated at cone 016)	20	20	20	20
Teague kaolin[a] (raw)	51	35		
Davis Mountain kaolin (350°C)			35	35
Dolomite	4	4	4	4
Flint		20	20	15

[a] Obtained near Teague, Freestone County.

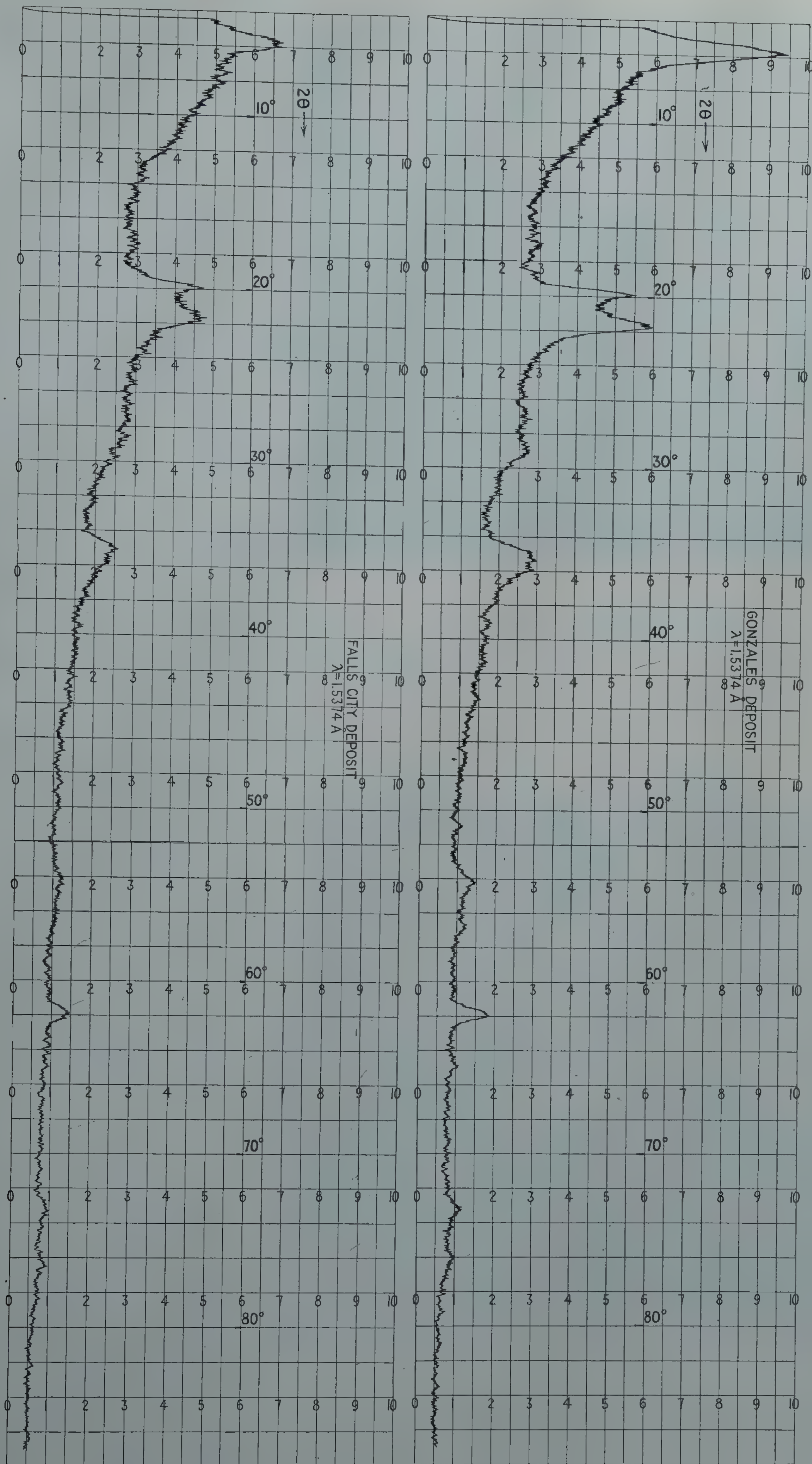

FIG 5—X ray diffraction patterns. Upper half, Gonzales; lower half, Falls City.

FIG 6—Gonzales. × 40,000.
(Reduced approximately one third)

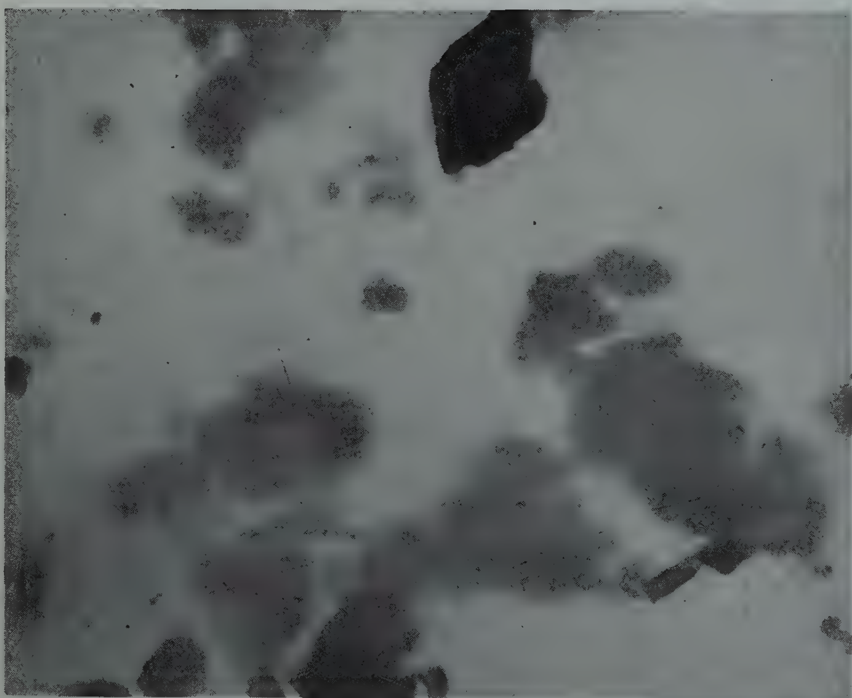

FIG 7—Falls City. × 40,000.
(Reduced approximately one third)

nated by the term thixotropic. Certain West Texas kaolins also possess this property in a slight degree, but when these kaolins are heated to 350°C this sticky property is overcome. A similar treatment, therefore, was tried on the bentonites but no change in physical property was noted. The author cannot, therefore, recommend the use of the bentonites in casting slips.

Only Texas raw materials were used in formulating semivitreous dinner ware bodies used in the laboratory experiments. Since the Gonzales bentonite possesses normal drying shrinkage, it was used for the raw bentonite content. Since Falls City bentonite vitrifies at Cone 5 and fuses at Cone 9, while at the same time showing excessive drying shrinkage, it was decided to use this bentonite only in calcined form as a substitute for feldspar. The bodies were formulated, fired at Cone 8, and shrinkages and absorptions obtained as shown in Table 4.

It will be noted that the shrinkages obtained are excessive, but that the absorption is satisfactory for semivitreous ware. It should also be noted that casting slips made from the raw body compositions above were unsatisfactory because of their thixotropic property.

Another series of bodies was prepared in which the raw bentonite is practically eliminated and the Falls City bentonite used was calcined at Cone 016 instead of Cone 06. Table 5 shows the body compositions used in these tests.

All of these bodies displayed satisfactory properties, both raw and fired, for semivitreous wares. Falls City bentonite calcined at the lower temperature gave satisfactory results, a fact which renders it more available as a substitute for feldspar.

Conclusions

White firing bentonite has a limited application as an ingredient in ceramic body mixtures, particularly in the raw state. The low vitrifying white bentonite when calcined at Cone 016, can be substituted for feldspar as a body ingredient in vitreous or semivitreous porcelains. White firing bentonite in the raw state has a definite ceramic application as a suspending agent in enamel slips. The white firing quality makes its use especially appropriate in white firing enamels.

Application of Screening and Classification for Improved Fine Anthracite Recovery

By W. J. PARTON,* Member AIME

THE efficient recovery and preparation of small sizes of anthracite called No. 4 Buckwheat (3/32 by 1/32 in.) and No. 5 Buckwheat (1/32 in. by 0), present a difficult problem to the anthracite operators. In many instances preparation of these sizes, particularly No. 5 anthracite, is extremely inefficient. In many of the older preparation plants, inadequate facilities, and often none at all, were provided as an integral part of the plant for cleaning this fine anthracite. Consequently, much of the material smaller than 3/32 in. was discarded. As the demand for these sizes increased, facilities for preparing the fine sizes were often crowded into existing structures with consequent sacrifices of capacities and efficiency. In addition to this condition, the methods of preparing the finest sizes are being developed in an attempt to improve their efficiencies. Because the demand for small sizes of anthracite still is increasing, many operators would benefit by investigating and improving the cleaning equipment employed on these sizes.

The object of this paper is to describe a stationary screen device from which the underflow is restricted by orifices, called a launder screen, along with operating results of several applications of this screen; a settling tank in which two classified products are produced by the installation of a cylindrical partition of smaller diameter than the original tank, one product being recovered from the small tank created by the partition and one product from the space between the inner and outer tank walls; a small pocket classifier or hydraulic trap for use in removing high ash particles from a sludge flow; the classification or segregation of sludge solids occurring in a 16 in. diam pipe flow by analysis of samples procured at different zones; and the operating results of a Fahrenwald sizer, as well as the performance of the concentrating tables handling the classified products. These simple devices were employed at the collieries of the Lehigh Navigation Coal Company Incorporated, Lansford, Pa., in order to improve preparation results on the fine sizes of anthracite.

San Francisco Meeting, February 1949.

TP 2532 F. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before April 30, 1949. Manuscript received Sept. 22, 1948; revision received Nov. 15, 1948.

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Launder Screens

The launder screen is a screening and dewatering device which probably could be used advantageously in the flow diagram of many fine coal plants. Fig 1 shows that it consists of a stationary screen constructed by placing 6 in. high partitions every 6 in. along a launder. Holes are then drilled on 6-in. centers across the bottom of the launder to receive pipe bushings of the desired diameter. The screen cloth is tacked on top of the partitions.

When water and solids are fed to the screen, only a small portion of the total water is removed through the orifices in each compartment so that the water in the feed is distributed evenly over the whole length of the screen. The screening action which results is very efficient because the solids are kept in a fluid condition for the full length of the screen. Blinding of the screen can be minimized by the use of screen cloth

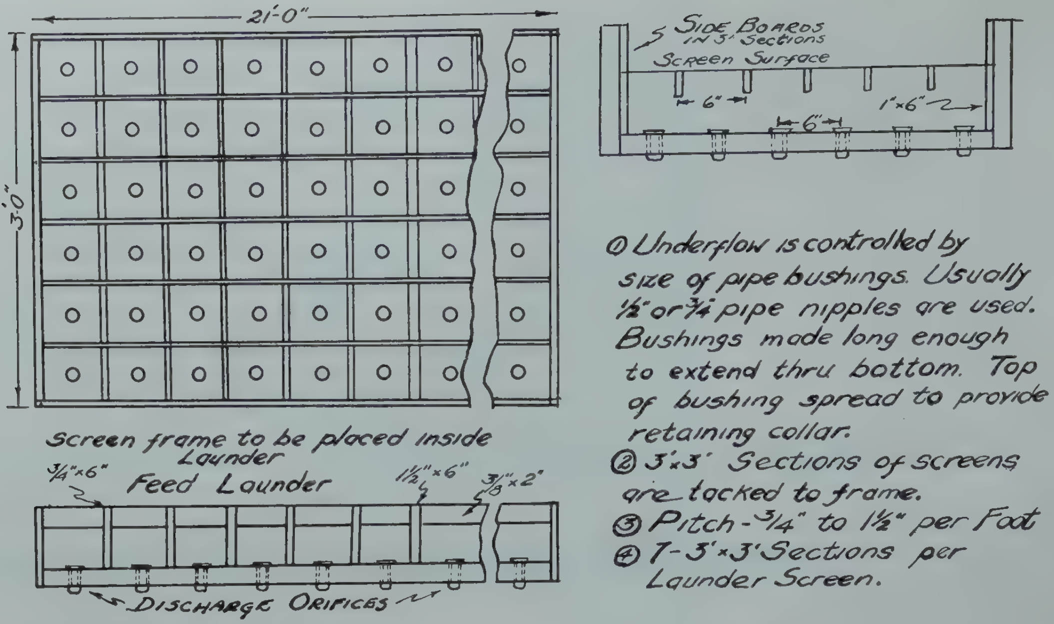

FIG 1—Launder screen.

Table 1 . . . Operating Results of Original No. 5 Anthrafine Plant, Nesquehoning Colliery

Sample Number	1		2		3		4	
	Feed		Refuse from No. 5 Cleaner		Underflow Dewatering Screen		Coal Product	
Tons Per Hour Ash, Per Cent	60 36.06		9.4 65.5		45.1 32.7		5.5 13.79	
Size Analysis	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
+8 mesh	0.00		0.32	53.6	0.00	10.05	0.20	8.3
+10	0.88	23.42	2.40		0.20		5.00	
+14	2.40	14.7	8.28	50.9	1.64		20.32	8.7
+20	26.56	29.0	35.16	62.1	17.60	14.1	39.48	11.7
+28	25.48	32.8	26.36	64.7	22.92	22.0	20.36	16.1
+35	18.80	39.9	15.56	69.6	20.92	34.3	9.60	25.6
+48	10.92	46.7	7.00	70.5	13.72	45.1	3.28	29.5
+65	6.12	60.4	2.96	75.7	8.52	56.0	1.20	34.2
+100	4.20	60.4	1.36	76.1	6.36	59.5	0.40	33.9
+150	1.68	57.3	0.08	63.8	2.64	59.6	0.08	
+200	1.08	52.5	0.40		1.80	54.0	0.04	
−200	1.88	51.5	0.12		3.68	56.0	0.04	
	100.00		100.00		100.00		100.00	

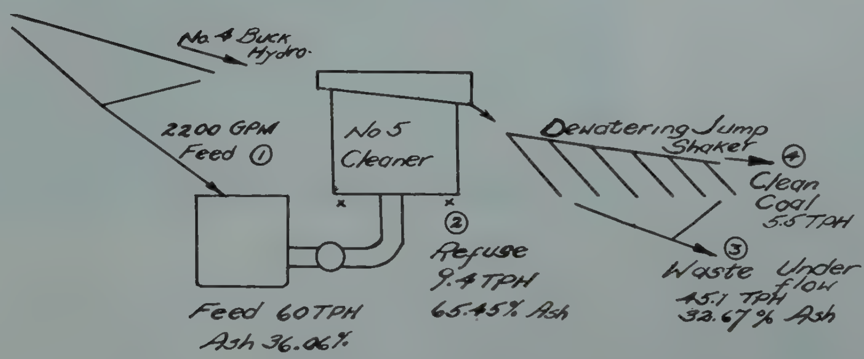

FIG 2—Original No. 5 anthrafine cleaning system, Nesquehoning Breaker.

Table 2 . . . Operating Results of Revised No. 5 Anthrafine Plant, Nesquehoning Colliery

Sample Number	1		2		3		4		5		6		7	
	Feed to Stationary Screen		Underflow or Stationary Screen		Overflow of Screen Feed to Cleaner		Cleaner Refuse		Underflow of 6 by 6 Ft Stationary Screen (Coal End)		Underflow Dewatering Screen		Coal Product	
Tons Per Hour	61		10		51		10		2.4		8.8		29.8	
Ash, Per Cent	39.0		52.83		34.18		71.83		48.55		46.43		14.74	
Gallons Per Minute	2,300		1,200		1,100				500		600			
Size Analysis	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
+10 mesh	0.2	19.9	0.0		0.9	23.7	2.1	37.1			0.1		0.2	7.8
+14	0.9		0.1	33.0	3.2	11.5	4.8	43.5			0.1	19.54	3.0	
+20	12.5	21.5	1.4		24.7	29.2	35.4	71.5			1.3		22.2	11.2
+28	18.5	20.7	2.0	35.3	25.8	27.1	29.1	74.2	0.2	31.4	4.0	19.23	40.92	11.8
+35	20.8	24.2	7.5	39.0	21.0	32.4	18.9	74.0	7.2		18.2	31.2	24.92	19.8
+48	15.0	32.6	16.5	47.9	11.4	38.2	6.3	72.9	23.4	41.5	22.5	47.1	6.80	41.8
+65	10.4	47.5	20.4	53.1	5.6	52.3	2.0	71.4	24.4	48.1	16.7	57.6	1.60	44.1
+100	8.5	57.3	20.9	57.8	3.1	58.7	0.9	71.1	20.9	54.8	13.5	56.8	0.20	
+150	4.4	58.1	10.7	60.7	1.3	56.7	0.2		9.4	57.4	7.2	56.0	0.08	
+200	3.3	59.0	6.0	58.5	0.8	54.1	0.1	66.7	4.7	55.8	4.5	54.6	0.04	41.6
−200	5.5	60.6	14.5	60.7	2.2	60.5	0.2		9.8	54.5	11.9	60.9	0.04	
Solids, per cent	7.92		3.33		17.4				1.92		4.36			

Note
3 ft of 24 mesh
15 ft of 20 mesh

Note
18 ft of 24 mesh

with approximately 50 pct, or more, open area and by the use of the proper orifices in the bottom of the trough. Large orifices cause more blinding of the screen.

Applications of this type screen are varied; it can be employed to remove excess water from a feed product ahead of cleaning equipment, as a screen to remove fine particles high in ash content from a sludge, or to produce a sized product. Examples of where it has been employed successfully are as follows:

Nesquehoning Breaker fine coal plant for No. 5 Buckwheat (1/32 in. by 0) was extremely inefficient because of the inadequacy of the hydraulic classifier type machine to handle the high ash feed. This classifier called a gyrotator is similar to a hydrotator except that the water is pumped through a stationary casting in the bottom of the tank. The casting is designed to impart a tangential flow to the water. Most of the product from the classifier was wasted in an effort to get a small tonnage to market at a satisfactory ash content. Fig 2 shows the flow diagram of the cleaning system previous to alterations and Table 1 gives the operating data.

In order to improve the efficiency of

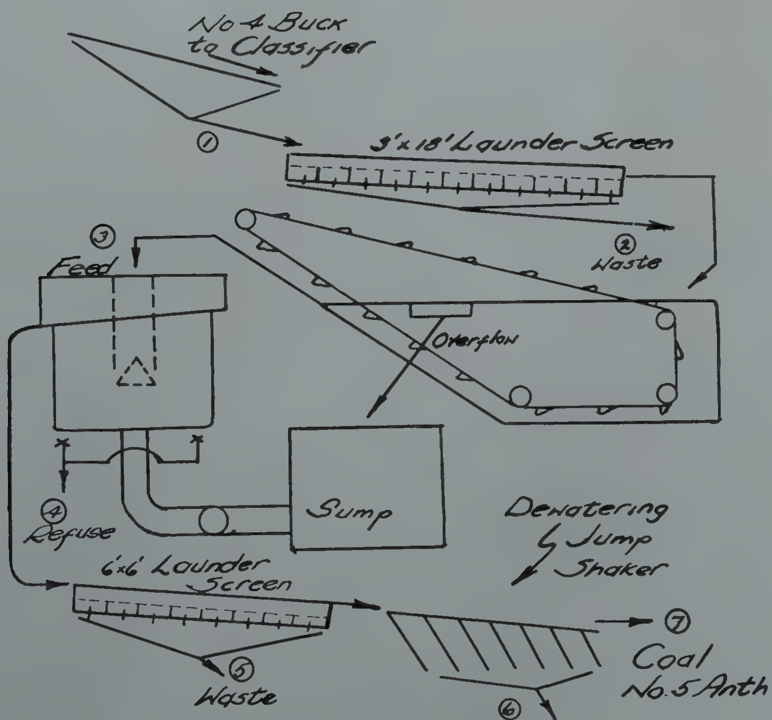

FIG 3—Revised No. 5 anthrafine cleaning system, Nesquehoning Breaker.

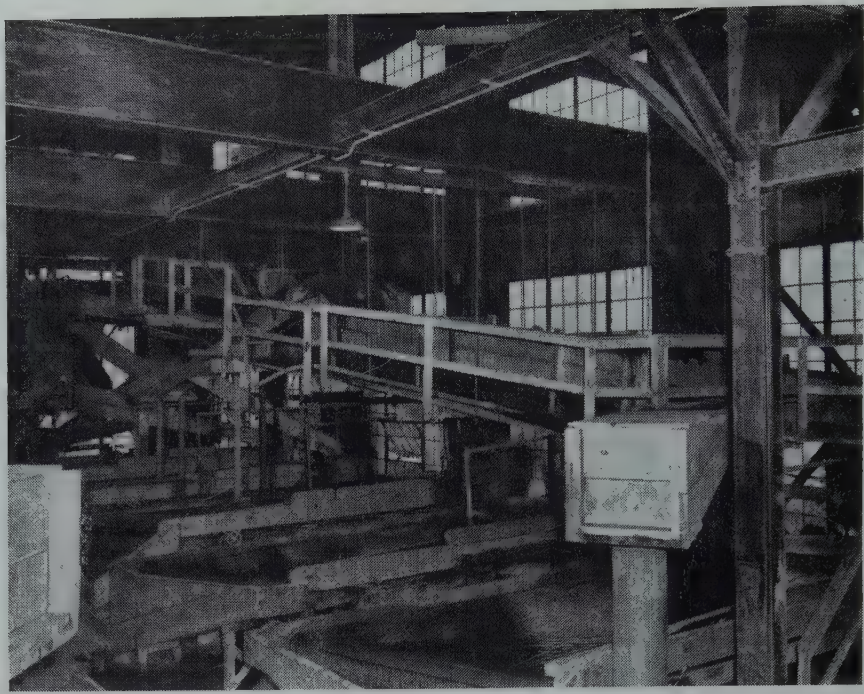

FIG 4—Installation of launder screens, Lansford Colliery.

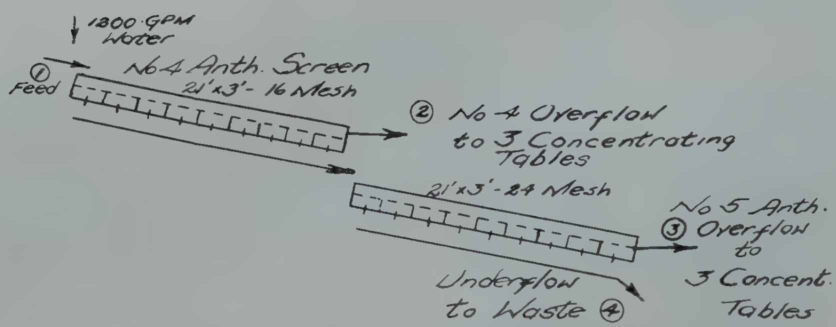

FIG 5—Launder screen circuit, Lansford Colliery.

Table 3 . . . Operating Results of Parallel Screening Circuit, Lansford Colliery

Sample Number Product Name	1 Feed		2 No. 4 Anthrafine Feed		3 No. 5 Anthrafine Feed		4 Underflow 24 Mesh to Waste	
Gallons Per Minute							900	
Solids, Per Cent							7.17	
Tons Per Hour	60.0		29.8		32.0		16.2	
Ash, Per Cent	28.18		28.68		34.09		46.79	
Size Analysis	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
+20 mesh	35.8	21.68	66.0	27.57	0.6	22.33	0.0	0
+30	19.4	23.66	20.8	29.23	18.6	29.11	0.0	0
−30	44.8	35.34	13.2	33.05	80.8	36.20	100.0	46.7

the cleaning system several changes were made by employing launder screens and a dewatering tank. The revised circuit is shown on Fig 3, and operating data in Table 2.

In this particular cleaning circuit the launder screen enabled 1200 gpm of excess water to be eliminated in the feed slurry and the removal of 10 tons per hour (tph) of high ash material from the circuit ahead of the cleaner. The second launder screen installed on

Table 4 . . . Operating Results, Lansford Launder Screen

Retained On	Feed to 16 Mesh Screen: Elevator Solids, Per Cent	Feed to 16 Mesh Screen: Dilution Water Solids	Oversize No. 4 Feed 16 Mesh Screen, Per Cent	Undersize 16 Mesh Screen, Per Cent	Oversize No. 5 Feed 20 and 24 Mesh Screens, Per Cent	Undersize to Waste 20 and 24 Mesh Screens, Per Cent
+8 mesh	0.08	0.0	0.20	0.00	0.00	0.00
+10	8.92	0.0	16.88	0.00	0.00	0.00
+14	6.48	0.0	16.40	0.00	0.00	0.00
+20	25.48	0.0	42.20	1.20	2.00	0.10
+28	18.80	0.0	15.08	16.00	29.36	2.30
+35	15.52	0.6	3.52	20.80	36.16	6.70
+48	9.72	0.8	1.08	14.20	14.68	11.50
+65	5.80	3.6	0.60	8.80	5.52	12.50
+100	1.20	10.8	0.40	6.50	2.68	5.00
+150	4.60	8.8	0.68	7.20	2.08	17.90
+200	1.40	8.0	0.68	5.50	1.48	7.60
−200	2.00	67.4	2.28	19.80	6.04	36.40
Total	100.0	100.0	100.0	100.0	100.0	100.0
Composite, ash	24.75	42.81	22.86	34.26	24.03	38.91
Solids, per cent		5.83			32.11	8.84
Flow, gpm		1200.			300.	1030.
Tons per hour	60.0	17.5	30.5	47.0	24.2	22.8
Recovery, per cent	100		39.4	60.6	31.2	29.4

the coal product from the classifier served to eliminate approximately 500 gpm of water which the original dewatering shaker was not capable of handling because of the increased bed on this screen resulting from increased recovery. Also this screen assisted in removing the high ash fines, which were overflowed by the classifier as indicated by the analysis of the solids.

Modification of this anthrafine circuit resulted in recovery of approximately five times the tonnage originally sent to market. The stationary screens efficiently removed high ash particles without appreciable blinding. Life of the bronze screens was approximately 84 hr.

Screen cloth employed had the following specifications:

Mesh	16	20	24
Wire diameter, inches	0.018	0.016	0.0135
Opening, inches	0.0445	0.034	0.0282
Open area, per cent	50.7	46.2	45.8
Launder orifices	½ in. diam		

At Lansford Colliery, the No. 4 Buckwheat (3/32 by 1/32 in.) and No. 5 Buckwheat (1/32 in. by 0) products were originally cleaned in a combined feed product by twelve concentrating tables. Because of the large range in particle size of this feed, table efficiency was very low. Furthermore, increased feed tonnages caused by a greater feed to the cleaning plant necessitated that increased cleaning capacity be provided or that proper sizing of the feed into the separate products be accomplished. Also removal of some of the fine high ash particles ahead of the tables seemed desirable. Accordingly launder screens were installed for this purpose as shown in Fig 4. Two screening circuits operating in parallel were constructed to handle the total tonnage of anthrafine feed. The flow diagram of one of the parallel screening circuits is shown on Fig 5.

Water for screening is provided by pumping part of the overflow from the anthrafine settling tank to the screens. Considerable fine high ash material is thus added to the circuit. However, this objectionable material is removed by the 24 mesh screen and is discarded. Preliminary operating results of this circuit are given in Table 3.

Wash water containing 5.93 pct solids at 36.99 pct ash was used for screening. As a result, 18 tons per hour of these high ash solids were introduced into the circuit. If clean water was available, considerable improvement would result in the screening and subsequent cleaning circuits.

Table 4 gives additional operating results of the launder screen when used for producing a sized product.

A 16 mesh screen is used to affect a 28 mesh cut. The velocity of the material passing over the deck reduces the size of particles which pass through screen openings. The oversize product, or the No. 4 feed, carries 77.4 pct of the plus 28 mesh material in the feed. Undersize in the product amounts to 9.24 pct. Overall screening efficiency for this screen at an effective cut of 28 mesh is 85.9 pct.

The secondary screen was dressed with approximately 70 pct 20 mesh cloth and 30 pct 24 mesh cloth at the time the samples were procured, for which analyses are given in Table 4.

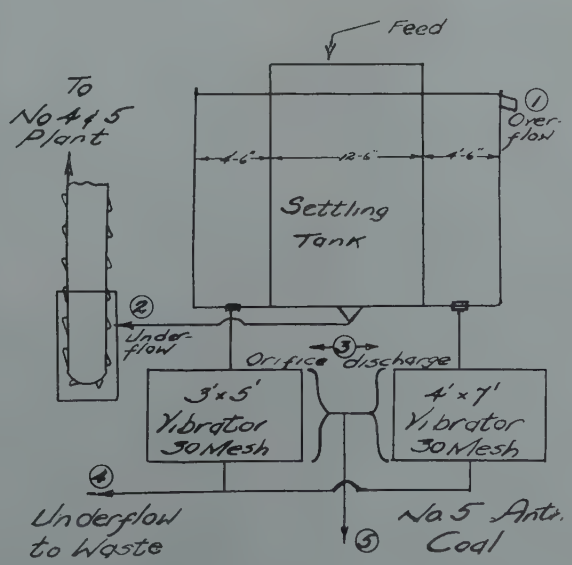

FIG 6—Settling tank circuit, Coaldale Colliery.

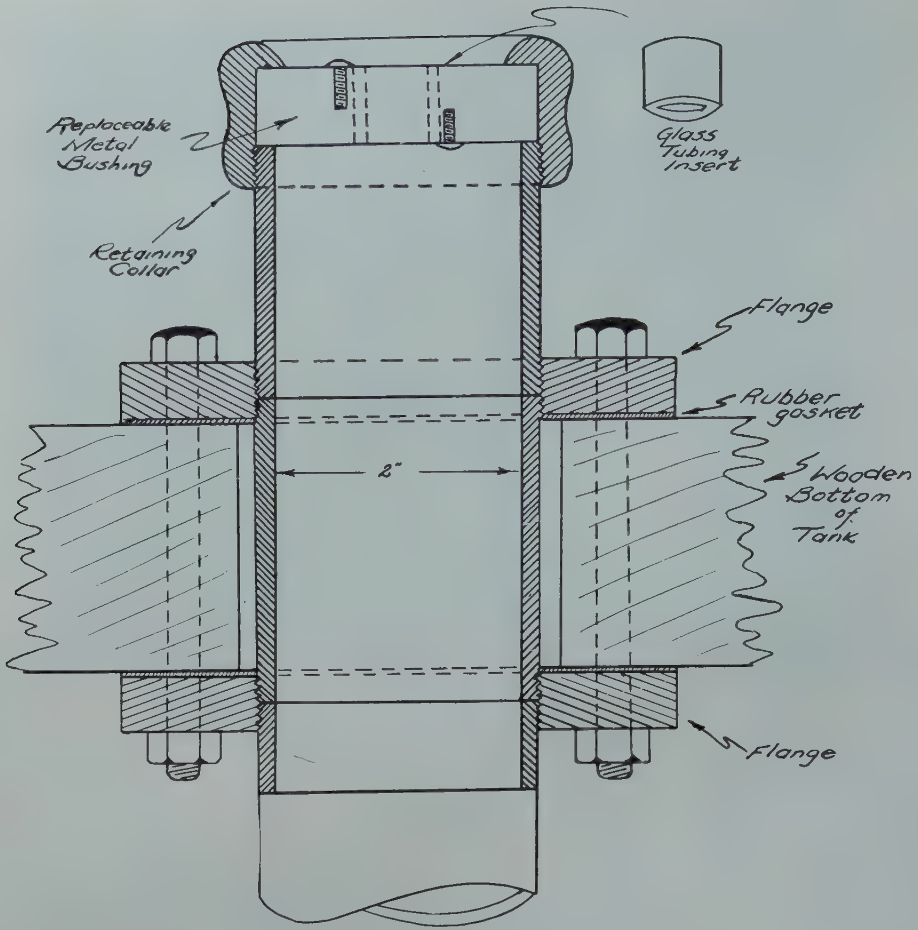

FIG 7—Orifice design.

For an effective separation of 48 mesh, the oversize product from this deck carried 81 pct plus 48 mesh particles. Overall screening efficiency at 48 mesh is 80.7 pct. The undersize product from this screen carried more plus 48 mesh material than desirable because of the use of some 20 mesh screen. A more desirable separation is made with all 24 mesh cloth on this secondary screen as indicated by the data in Table 3. A complete size analysis of a 24 mesh cloth underflow is shown in Table 5.

When used for sizing, the flow over the launder screen should be regulated so that essentially all the water has been removed through the cloth by the time the flow reaches the discharge point. Only sufficient water should be allowed to overflow the end of the screen to provide continuous removal of the oversize particles. When dirty water is used for screening, a good practice is to remove all this water through the screen several feet from the end of the screen. Sufficient clean water should be added to this point to flood the oversize particles over the remaining length of the deck.

For comparison with the bronze screen cloth, a stainless steel wire cloth was recently placed in service. Longer life and less blinding is obtained with this screen, as compared with the bronze cloth. Minimum life of the stainless steel cloth was 450 hr. Most of the cloth originally installed is still in service at the writing of this paper.

One of the problems encountered with the launder screen is blinding from a hair or fiber-like material, which requires the running of a steel brush over the deck occasionally. This material may be wood fiber, excelsior, or hay. The use of fresh water for screening rather than dirty, recirculated water will undoubtedly overcome this difficulty to a great extent.

Table 5 . . . Size Analysis of 24 Mesh Cloth Underflow

Retained on	Material, Per Cent	Ash, Per Cent
35 mesh	0.7	30.11
48	5.3	
65	13.6	41.98
100	16.5	
150	11.6	41.59
200	8.0	
−200	44.3	54.62
Composite	100.0	46.79

Coaldale Colliery Anthrafine Plant

Recovery of No. 4 Buckwheat ($\frac{3}{32}$ by $\frac{1}{32}$ in.) and No. 5 Buckwheat ($\frac{1}{32}$ in. by 0) at Coaldale Breaker was made by settling the raw feed (representing the $\frac{3}{32}$ in. underflow from the main sizing shakers) in a 12 ft 6 in. settling tank. This settling tank was originally 21 ft 6 in. in diameter but was reduced in size because of inadequate markets and difficulty in cleaning the finest particles. With the increasing demand for fine sizes of anthracite created during World War II, additional tonnages of classified feed were recovered by

Table 6 . . . Coaldale Anthrafine Settling Tank Operating Results

Sample Number	1		2		3		4		5	
	Final Overflow Tank, 21 Ft 6 In.		Underflow of 12 Ft 6 In. Tank Feed to Original Anthrafine Plant		Underflow of Outer Tank through Orifices Feed to Screens		Underflow 30 Mesh Screens		Coal Product Screen Oversize	
Flow, GPM	5000									
Solids, Per Cent	6.04									
Solids, TPH	78		101		45		15		30	
Ash Content, Per Cent	33.60		29.98		24.72		35.96		14.69	
Size Analysis	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
+8			2.0	20.2					0.1	11.9
+10			16.0	22.2	0.2	10.3			0.9	
+14			14.4	17.5	0.5				1.0	8.2
+20			23.6	28.9	3.5	9.4	0.1		8.5	9.2
+28	0.4	17.9	18.1	29.0	10.3	9.1	0.2	15.44	22.4	9.2
+35	1.6		11.1	38.9	17.8	12.0	5.3		33.8	12.6
+48	4.0	15.9	5.8	46.5	17.7	17.8	15.4	20.9	18.2	18.7
+65	6.4	14.9	3.0	50.4	14.5	25.1	19.0	29.0	7.8	24.3
+100	13.6	20.2	2.3	49.7	13.2	34.4	21.2	39.0	4.2	33.7
+150	10.8	25.21	1.3	42.3	7.4	40.4	13.5	45.3	1.3	38.2
+200	10.4	29.54	0.8	39.9	3.4	40.7	7.0	44.8	1.0	37.2
−200	52.8	43.5	1.6	41.8	11.5	45.5	18.3	47.0	0.8	36.6

using the space between the inner and outer tanks. Material which settles in this area is discharged through a series of orifices placed in the bottom of the tank. Fig 6 shows the design of the settling tank and the position of the underflow orifices. The flow diagram also shows that the classified solids recovered through the orifices are screened over two vibrating screens to remove the high ash fines and thus produce a satisfactory product for market.

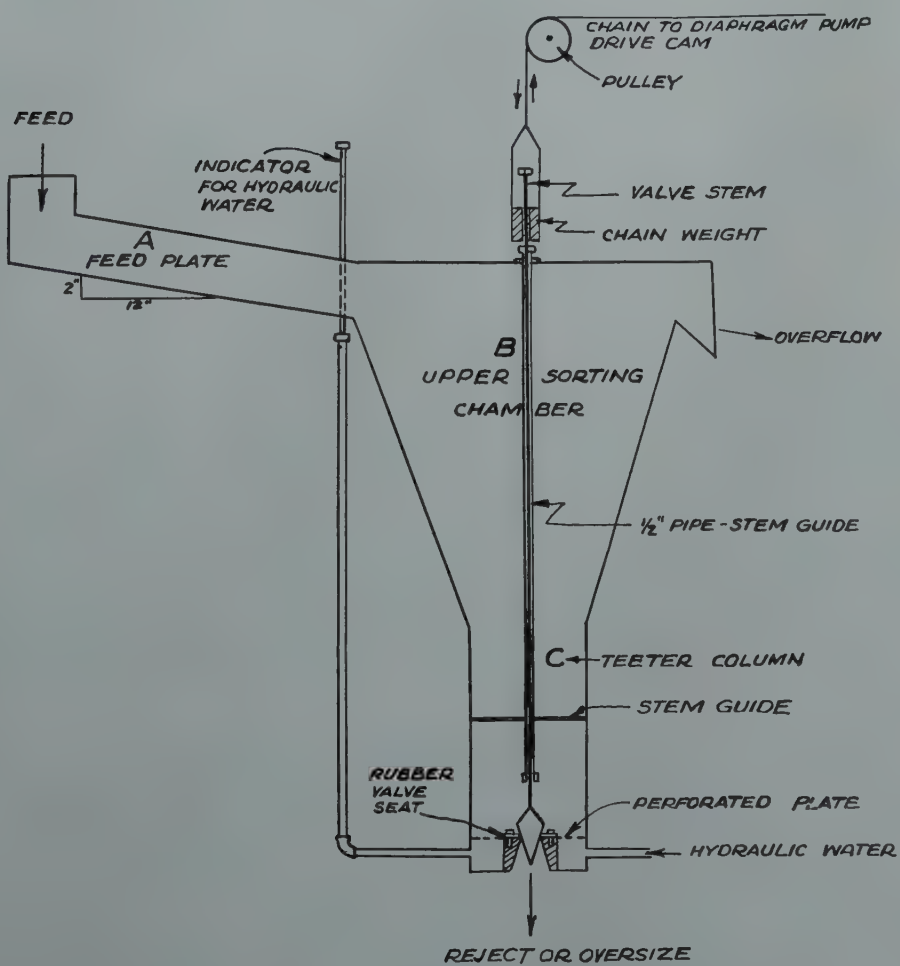

FIG 8—Oversize classifier.

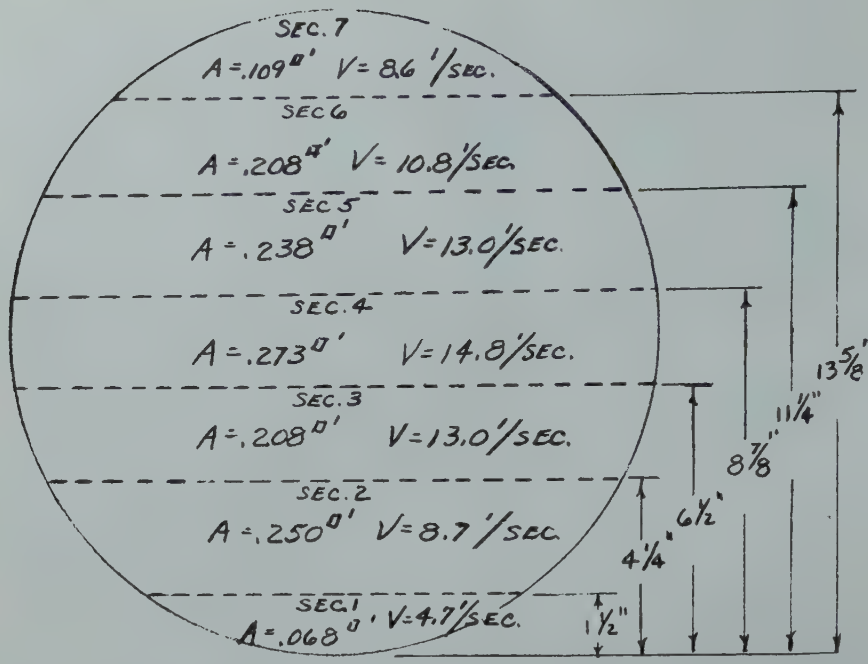

FIG 9—Wood pipe (15¾ in. diam) sections sampled and analyzed for classification test.

Operating results of this circuit are given in Table 6.

Several types of replaceable orifices were used in the bottom of the tank to regulate the discharge. Fig 7 shows one type of orifice with a replaceable glass insert held in place by the bolt heads. Life of this orifice is approximately one month as compared with two weeks for a plain steel orifice. Twenty-five orifices were installed in the tank bottom to obtain the desired discharge flow.

The recovery of additional No. 5 anthrafine by discharging the classified material through orifices in the bottom of the tank and screening this material to eliminate high ash fines resulted in an increase of approximately two and one-half times the original tonnage.

Pocket Classifier

A small pocket classifier designed for use in treating the oversize product of a flotation plant has proved to be of considerable help to remove pyrite and high ash particles from the underflow of a hydroclassifier. Fig 8 shows the design of this machine.

The classifier is of the surface current type consisting of a feed plate *A*, upper sorting chamber *B*, and the lower sorting chamber or teeter column *C*. Hydraulic water is introduced at the bottom of the teeter column by diffusing it through a perforated screen. A stem-type valve is used to control the discharge of refuse or classified oversize from the bottom of the teeter column. The cam action of the diaphragm pump was used to raise and lower the stem valve to permit the desired underflow discharge by attaching a chain through pulleys from the cam to the top of the valve stem. The degree of opening of the valve on each stroke (45 rpm) is controlled by the amount of slack in the chain. This type of classifier shows promise for use in preparing classified feed for concentrating tables. When used for removal of high ash particles from a feed of 140 gpm at 32 pct solids, a typical underflow sample from an 18 by 24 in. classifier is given in Table 7.

Table 7 . . . Typical Underflow Sample

Underflow, gpm	11
Solids, per cent	40
Tons per hour	1.1
Ash, per cent	58.41

Size Analysis	Weight, Per Cent	Ash, Per Cent
+6	0.9	48.34
+8	1.3	
+10	4.2	
+14	7.6	
+20	23.4	66.65
+28	18.8	
+35	14.8	63.3
+48	12.5	
+65	8.4	74.0
+100	5.4	
+150	1.7	67.0
+200	0.6	
−200	0.4	57.5

In the operation of the classifier, primary stratification takes place according to size and gravity as the pulp flows over the feed sole or plate. The

Table 8 . . . Results of Pipe Classification Test

Section	Velocity Ft per Sec	Discharge, GPM	Solids, Per Cent	Solids, TPH	Solids +28 Mesh, TPH	Solids +35 Mesh, TPH
1	4.7	145	17.8	7.0	3.4	4.5
2	8.7	977	7.0	17.5	3.3	5.9
3	13.0	1,220	5.0	15.5	1.3	2.8
4	14.8	1,830	4.4	20.6	0.8	1.6
5	13.0	1,400	3.7	13.0	0.2	0.6
6	10.8	1,020	3.0	7.7	0.1	0.3
7	8.7	425	2.5	2.8		
Composite	11.5	7,000	5.36	95		

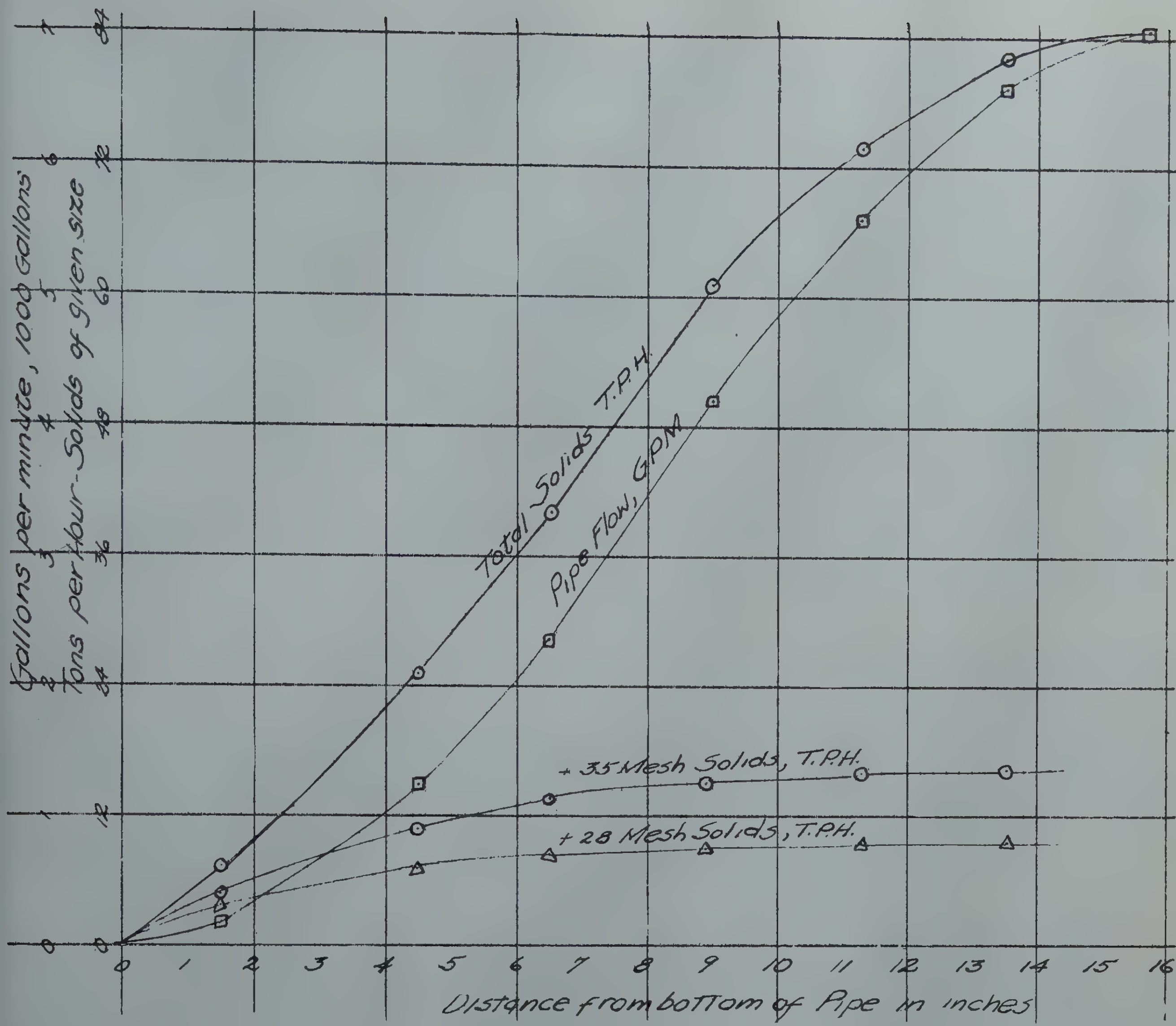

FIG 10—Flow characteristics of 16-in. pipe.

partially stratified particles enter the upper sorting chamber which sloughs off the lightest and finest particles. Final separation on the particles which settle in the upper chamber is accomplished in the teeter column.

Classification in Pipe Flow

A test was conducted on the flow of silt-laden water from the 16-in. wood pipe line feeding the Tamaqua Colliery froth flotation plant to determine the amount of segregation and classification of solids that occurs. Approximately 7000 gpm of slurry water was passed through 400 ft of pipe. Sampling of the line was done by introducing a slicer at different elevations in the cross section of the pipe and procuring samples from each section. The velocity of each pipe section was approximated relatively by measuring the rate of discharge through the slicer when placed in the different pipe zones to procure the samples. Fig 9 shows the manner in which the cross section of the pipe was divided. A summary of the results from several tests is given in Table 8.

Fig 10 shows graphically the distribution of the flow and solids in the cross section of the flow. The lower half of the pipe flow carries approximately 8.4 tph of the plus 28 mesh particles in 3200 gal of water. The upper section carries only 0.7 tph of the plus 28 mesh material.

Table 9 gives the size and ash analyses of the solids recovered in the various cross-section zones of the pipe except for Zone 7 which was not sampled on this particular test. These data indicate that considerable classification of the suspended particles takes place in the pipe flow. This condition makes possible the splitting of the flow into separate zones to enable the separate handling of that part of the flow which carries the preponderance of the desired particles.

Fahrenwald Sizer

The advantages obtained from improved table operation by feeding the tables with a classified feed have been discussed much in the past. The use of a Fahrenwald sizer on an anthrafine feed consisting of both No. 4 anthrafine (3/32 by 1/32 in.) and No. 5 (1/32 in. by 0) sizes will be described by data in Tables 10 and 11. Data on sizer performance in Table 10 indicates that Spigots 1 and

Table 9 . . . Pipe Classification of Feed Solids, Tamaqua Flotation Plant

Marked	16 In. Pipe Bottom Section 1		16 In. Pipe Section 2		16 In. Pipe Section 3		16 In. Pipe Section 4		16 In. Pipe Section 5		16 In. Pipe Section-top 6		Composite Pipe Sample	
Flow, GPM	145		977		1,220		1,830		1,400		1,020		7,000	
Solids, Per Cent	17.25		8.23		4.67		3.8		3.5		3.14		5.36	
Tons Per Hour	7.0		20.6		14.6		17.7		12.5		8.10		95	
Composite Ash	24.85		31.97		33.28		34.50		36.94		38.21		32.25	
Size Analysis	Material, Per Cent	Ash, Per Cent	Material, Per Cent	Ash, Per Cent	Material, Per Cent	Ash, Per Cent	Material, Per Cent	Ash, Per Cent	Material, Per Cent	Ash, Per Cent	Material, Per Cent	Ash, Per Cent	Material, Per Cent	Ash, Per Cent
+6	0.1		0.0		0.0		0.0		0.0		0.0		0.0	
+8	0.2	25.44	0.0		0.0		0.0		0.0		0.0		0.1	
+10	2.6		0.3		0.0		0.0		0.0		0.0		0.4	
+14	2.7		1.2	16.03	0.2	10.86	0.2	19.83	0.2		0.2		1.0	18.00
+20	22.0	16.59	8.5		2.3		1.2		0.2	13.26	0.2	14.23	6.5	
+28	19.5		14.0		5.0		2.6		1.4		0.6		10.7	
+35	15.0		16.2		9.7		2.4		3.4		2.0		12.5	
+48	10.2	28.41	14.8	33.36	11.0	24.53	13.0	21.33	5.4	14.51	4.0	12.74	11.5	32.48
+65	7.8		12.2		10.7		10.0		6.6		6.0		11.0	
+100	7.0		11.0		12.2		13.0		9.0		9.0		10.0	
+150	3.5	41.86	5.0	40.14	7.5	36.30	8.4	33.12	7.0	28.13	6.8	26.50	5.5	36.27
+200	2.5		2.5		5.0		6.0		6.0		6.0		4.0	
−200	6.9	43.57	14.3	53	36.4	46.15	43.2	46.67	60.8	47.94	65.2	47.19	26.8	43.44
Total	100.0		100.0		100.0		100.0		100.0		100.0		100.0	

Table 10 . . . Fahrenwald Sizer Operating Results

Mesh	Feed to Sizer		Feed Table No. 1 Spigots 1 and 2		Feed Table No. 2 Spigots 3 and 4		Feed Table No. 3 Spigots 5 and 6		End Overflow		Side Overflow	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
+10	2.6	22.5	0.8	23.3	1.0	10.36	0.0					
+14	6.1	22.8	1.7	30.1	5.7	9.20	0.1					
+20	12.4	22.8	3.1	34.2	18.1	11.6	1.2	7.62				
+28	18.0	24.7	6.1	43.3	31.8	15.15	10.1	7.13				
+35	18.0	26.9	13.1	60.9	24.3	27.23	27.5	8.75	2.6	8.0		
+48	13.7	30.3	23.9	74.8	7.0	49.25	29.0	10.05	15.5	7.35	4.1	10.62
+65	10.5	34.2	27.3	82.5	6.2	72.65	16.6	28.09	26.3	10.61	15.3	14.52
+100	9.6	44.9	4.9	87.3	4.1	86.05	10.3	68.94	25.3	23.17	27.0	32.42
+150	4.8	53.6	17.2	87.2	1.3	90.78	3.7	81.86	13.2	38.52	15.5	38.77
+200	1.8	52.7	1.5	72.0	0.3	76.60	0.9	85.39	5.8	54.29	8.7	50.0
−200	1.2	58.1	0.4		0.2		0.6	75.46	11.6	51.15	29.4	53.24
Composite												
Ash, per cent		29.05		43.46		30.41		24.05		24.87		34.76
Solids, per cent	45.7		50.0		48.0		30.0		9.9		6.7	
Long tons per hour	26.3		7.0		7.7		7.3		0.2		4.1	

Table 11 . . . Table Performance on Classified Feed

Mesh	Table No. 1 Spigots 1 and 2				Table No. 2 Spigots 3 and 4				Table No. 3 Spigots 5 and 6			
	Coal		Refuse		Coal		Refuse		Coal		Refuse	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
+10	9.5	11.57	1.6	79.41	1.1	8.40	0.1	33.59	0.7	9.41		
+14	25.5	11.88	7.2	79.74	8.4	7.80	0.8	38.62	2.1	10.83		
+20	39.3	11.51	16.1	74.39	27.3	8.59	3.8	39.2	5.1	12.70	1.3	65.82
+28	21.3	11.90	27.7	65.20	39.5	9.59	16.4	44.66	14.3	11.37	1.3	74.15
+35	3.1	13.27	24.0	67.38	18.4	12.01	31.9	61.08	29.5	10.79	2.5	66.60
+48	0.6	29.14	12.2	75.90	4.0	16.61	23.1	75.90	28.3	13.57	5.7	52.87
+65	0.9	54.91	6.2	81.67	0.8	27.07	12.8	87.73	13.3	20.66	12.9	48.65
+100			3.7	88.46	0.2	53.40	8.2	86.92	5.3	39.49	23.7	59.28
+150			1.0	88.76	0.3	60.14	2.3	87.08	1.2	60.74	33.4	81.34
+200			0.3	78.56			0.4	72.67	0.3	72.39	14.2	88.53
−200							0.2	60.75	0.3	60.43	3.3	89.54
Composite											1.7	85.44
Ash, per cent		12.28		76.25		10.70		60.75		17.52		70.58
Long tons per hour	3.3		3.7		4.9		2.8		6.1		1.2	
Solids, per cent	17.3		53.4		16.4		48.7		No Data			

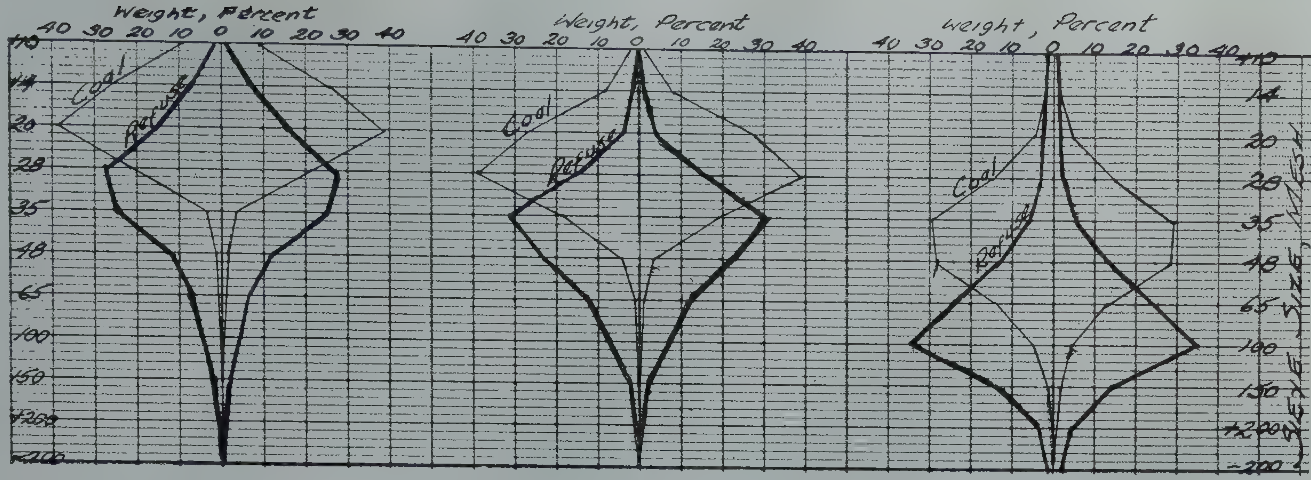

FIG 11—Table performance on classified feed.

2 discharged much of the high ash material resulting in a composite ash of 43.46 pct. Most of the material smaller than 28 mesh consisted of refuse particles. This product consists essentially of a No. 4 anthrafine feed. The coal end of the table operating on these spigot products, analysis in Table 11, shows that only 4.6 pct of the product is smaller than 28 mesh. The discharge from Spigots 3 and 4 consists of an in-between product of both No. 4 and No. 5 anthrafine feed, which when cleaned on a table produced a coal product consisting of 23.7 pct material smaller than 28 mesh. Table performance on both spigot products was good, with satisfactory coal and refuse products. The discharge from Spigots 5 and 6 consisted essentially of a No. 5 anthrafine feed with only 11.4 pct plus 28 mesh material present. Table performance on this product was not as good as on the previous products because part of the fine high ash particles reported to the coal product to build the ash content to 17.52 pct. However, by reducing the tonnage to this table and adjusting the table variables, a satisfactory product could have been obtained.

Fig 11 shows graphically by use of bilateral diagrams the difference in size of the refuse and coal products from the tables operating on the classified feed products from the Fahrenwald classifier. The bilateral diagram is a convenient method of showing visually the difference in size consistence of different products as explained by Coe, Feld, Williams and Coghill.*

This diagram is of particular interest because it shows clearly the sizing which takes place by both particle size and specific gravity. Due to equal settling ratios of the particles, large particles of coal and small particles of refuse report to the individual spigot products.

It is because of this difference in size of the low gravity and high gravity particles from a given spigot that the tables are able to perform efficiently on a classified feed.

The decreasing size of the products, both coal and refuse, discharged from the succeeding spigots can also be noted by comparing the various product curves.

Table performance can be improved by the preparation of classified feed to the tables. Control of the spigot discharge from the Fahrenwald sizer was done by a rubber diaphragm which actuated a plug valve in each classifying pocket of the sizer. A newer type of control utilizing a Pressuretrol and electric motor is more sensitive and makes possible the preparation of more uniform discharge products despite varying feed conditions as to quantity and character.

* G. Dale Coe, I. L. Feld, M. F. Williams, Jr. and Will H. Coghill: Continuous Hydraulic Classification: Constitution of the Teeter Column Throughout Its Depth. U.S. Bur. Mines, *R.I.* 3851.

Conclusion

The utilization of simple classification and screening devices have resulted in improved recovery of No. 4 and No. 5 anthrafine at the cleaning plants of the Lehigh Navigation Coal Company Incorporated. These devices have been described with operating results to promote further thought and interest on means of improving the preparation of the fine sizes of coal. Considerable study must be made on the preparation of finest sizes since the present equipment and circuits are none too efficient.

The launder screen is a tool that can be used to advantage in numerous fine coal cleaning circuits. The operating data procured when used to prepare a sized product indicate that a closely sized product can be attained at high screening efficiency.

Operational Statistics of a Marion 5560 Power Shovel

By GEORGE B. CLARK,* Member AIME, and GEORGE L. REITER†

Strip Mining

COMMERCIAL strip mining of coal was first begun in the state of Illinois in 1911.[1] The annual tonnage of coal produced from coal strip mines in the state was very small until 1924, when the strip mines shipped 1.68 pct of the total state shipping mine tonnage. Since then, strip mines have gradually come into their own by producing a sizable percentage of the coal produced in the state. Table 1 shows the coal tonnage shipped from both underground and strip coal mines in the state of Illinois for selected years. The table illustrates the major role that strip mines play in the state's coal industry.

Today, strip mining is generally conducted in areas where the coal seams lie so close to the surface of earth that a stripping method of mining the coal seam is cheaper than an underground method. Strip mining not only recovers 100 pct of the coal deposit, but may also be used as a method of mining large acreages of coal abandoned by the closing of underground mines.

Open cut or strip mining is a very highly mechanized method of producing coal and is carried on in three distinct and separate operations. The first of these is uncovering or exposing the coal seam so that the coal is made available for mining. The second operation is that of actually mining the coal. The third operation is the washing, crushing and grading of the coal according to the producer's market.

Description of Shovel

The Marion 5560 power shovel is used in the first of these three operations. The large electric-powered shovel works forward and back across the coal deposits, removing the overburden of dirt, rock, and clay in successive cuts of 30 to 60 ft deep and 40 to 80 ft wide. The shovel has a maximum cutting height of 95 ft, but around 60 ft is considered to be the maximum economical cutting height by the operators. The smaller the depth of cut, the more economically the shovel will operate until a minimum cut of 15 ft is reached, below which the efficiency decreases rapidly.

The Marion shovel may be used in tandem stripping in excessive depths of overburden. The other type of excavating equipment used in the tandem team is generally a dragline with a long boom of about 180 ft. In tandem stripping, the dragline is used to remove the upper portion of the cut and to place it at a maximum dumping radius from the place of removal. The Marion 5560 power shovel is then used to remove the overburden left above the coal seam and to deposit it at the shovel's maximum dumping radius.

The Marion 5560 power shovel is of gigantic proportions. It has a 102 ft 6 in. boom and 32 cu yd dipper and is

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† Student, University of Illinois.

[1] References are at the end of the paper.

mounted on 8 separate crawlers—2 crawlers at each corner of the lower frame. The crawler units at each corner of the frame are massive—the crawler belts being 42 in. wide and the overall crawler length is 19 ft. Since the weight of the shovel is over 1600 tons, when-

proving methods, eliminating delays, and developing time standards for future production. In many industries such studies are aimed only at appraising the quality of labor, but in general this type of study is made on highly standardized operations which are

1. Digging
2. Swinging and dumping
3. Return swing
4. Moving shovel
5. Delays

Table 1 . . . Coal Tonnage Shipped from Underground and Strip Coal Mines in Illinois[2]

Year	All Mines		Strip Mines Only			
	Total Tonnage Produced	Number of Mines	Number of Mines	Per Cent of Number Mines	Total Tonnage Produced	Per Cent of Total Tonnage
1946	60,932,785	160	36	22.5	14,302,739	23.47
1940	46,071,806	139	27	19.4	12,024,635	26.10
1935	41,410,414	182	28	15.4	7,088,104	17.11
1930	51,996,608	185	15	8.11	6,220,336	11.96
1924	70,324,363	338	11	3.25	1,184,288	1.68
1920	72,409,610	373	3	0.80	367,009	0.50

ever it is desired to move the shovel over a weak-surfaced strata, it is necessary to place large timber matts over the surface for the shovel to drive on.

There are two hoisting motors of 350 hp each that drive a single-reduction herringbone gear drive for the 65-in. hoisting drum. Power for rotating the shovel's upper frame is supplied from 3 motors of 125 hp each.

The shovel has dual control stations —one on each side of the boom. Thus the operator has excellent visibility of the operations at all times for either direction of swing or travel, and he may choose the side of boom from which he desires to work. Normally only two hand and two foot controls are used for digging, swinging, dumping, and return swing. A large control panel for other adjustments that may be required is placed in the center.

The massive dipper is built up of castings and plates welded together so they form as a single unit. The cutting edge of the dipper has seven large, replaceable cutting teeth weighing around 200 lb each.

Time Studies

The time study is one of the most useful tools of modern production management. If properly conducted, it not only provides the information necessary to set production standards, but furnishes the basic data for im- repeated many times under identical conditions. To be completely informative, however, time studies should yield information not only concerning: (1) the quality of labor, but also (2) the quality of machinery, and (3) the quality of working conditions.

Table 2 . . . Time Distribution for 1947

	Hour	Minutes
A. Overall operating time:		
Digging	5,584	03
Moving	444	46
Delays	2,155	11
	8,184	00
B. Productive time:		
Digging	5,584	03
Moving	444	46
	6,028	49
C. Delay time:		
Electrical failure	299	12
Mechanical failure	1,033	33
Power failure	78	41
Miscellaneous	743	45
	2,155	11

In some mining operations the various elements of a given operation may be classified as "constants" or "variables." In stripping operations, however, none of the elements of the operation cycle seem to fall into the class of "constants."

The analytical methods of time studies are very readily applicable to the operation of a large stripping shovel. For the purposes of such a study the operation may be reduced to the following elements:

The shovel is equipped with a device in the cab which records the time of these factors on a moving tape. The record is made in such a manner as to show the degrees of arc swung by the boom. Daily records are taken from the tape and are transferred to ledgers. From these ledgers monthly and yearly summaries are made. Table 2 is such a summary for the year of 1947. The shovel's entire operation period of 8184 hr has been broken down into (1) operating time and (2) delay time. These in turn have been separated into their elemental components.

ACTUAL STRIPPING TIME

Fig 1 to 3 show the time distributions from Table 1 in graphically illustrated form. These graphs show readily that during the year of 1947 a total of 8184 hr were credited to the shovel, but only 5584 hr, or 68.1 pct of the time, was used in actual stripping operations. Of the total operation time, 5.4 pct was employed in moving the shovel progressively to new adjacent operating positions. These figures yield an overall "efficiency" of 73.5 pct for the year of 1947. "Power failure" was that due to storms, and other reasons, outside of the company property. "Electrical failure" was that due to causes on company property.

The aggregate amount of moving involved was 114,816 ft. Because of its enormous size and weight, the shovel travels slowly. It normally requires

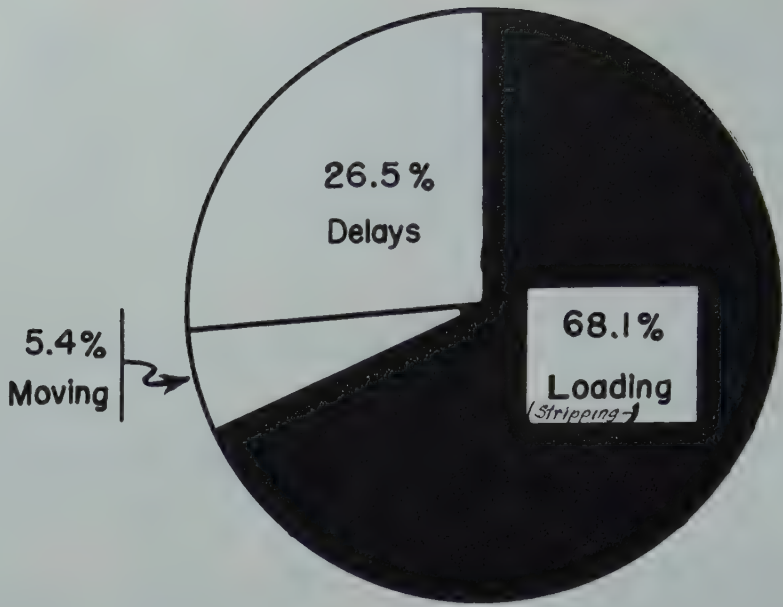

FIG 1—Overall time distribution.

about 3 min and 50 sec to travel 18 ft and relevel for digging.

Table 3 presents an average time distribution for an operating period of about 4 hr. Observed times were recorded from a stop watch and the degrees of swing were estimated. Fig 4 is a graphical representation of the average proportionate time obtained by this method.

Table 3 . . . Time Distribution for 102 Observed Dipper Cycles

	Per Cent
Swing	26.4
Dump and return swing	30.8
Digging	42.8
	100.0

Because of the fact that dumping and swinging operations overlap, it is very difficult to segregate the actual time required to dump the dipper. However,

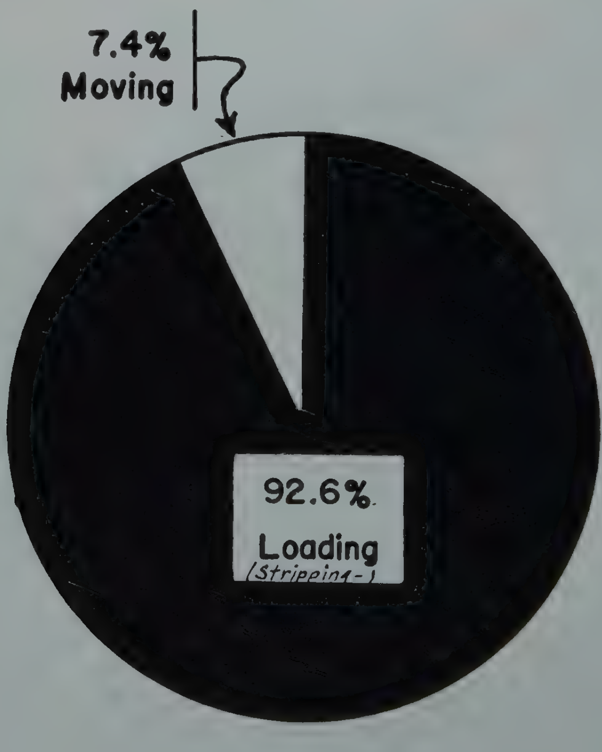

FIG 2—Time distribution for productive operations: moving, and loading and dumping.

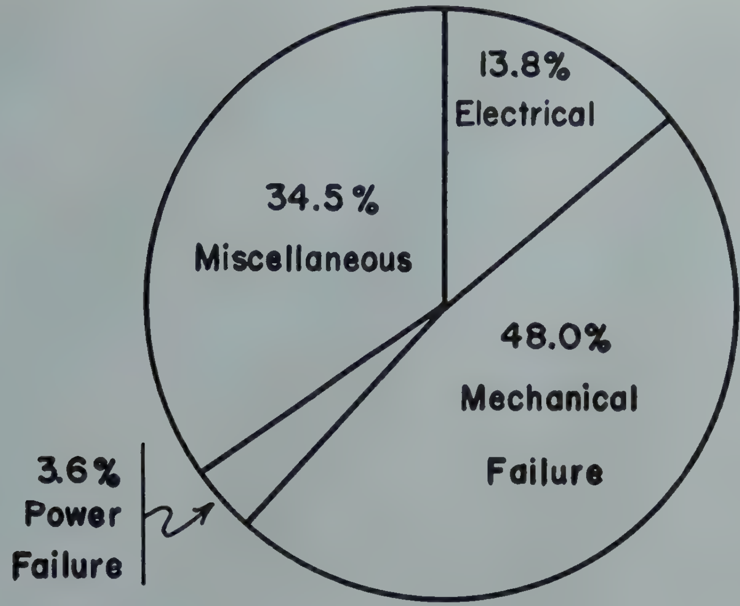

FIG 3—Time distribution of delays.

the time normally varied between 3 to 6 sec for the dipper to be dumped and the shovel to begin the reverse swing. The digging stroke time represented in Fig 4 is slightly higher than in the average digging stroke in stripping because of the fact that several cleaning strokes must be made each time the shovel is moved. The range of time for each element of the cycle varied between the following limits:

Dig	14 to 110 sec
Swing	11 to 27 sec
Dump and return swing	15 to 28 sec
Angle of swing	40° to 180° (one way)

The wide variation in the time consumed per digging stroke was caused by the variation in depth and hardness or compactness of the material, as well as by the fact that certain of the strokes were required for cleaning up.

In Fig 5 the time per cycle and the angle of swing are plotted as ordinates with the dipper number along the horizontal axis. As might be expected, there is not an exact correlation for each individual dipper cycle, but the general trends are parallel. That is, as the angle of swing increased, the time per dipper increased also. The swinging and dumping time is almost constant for a given angle of swing, the variation in time per dipper being almost wholly due to the variable time required to fill the

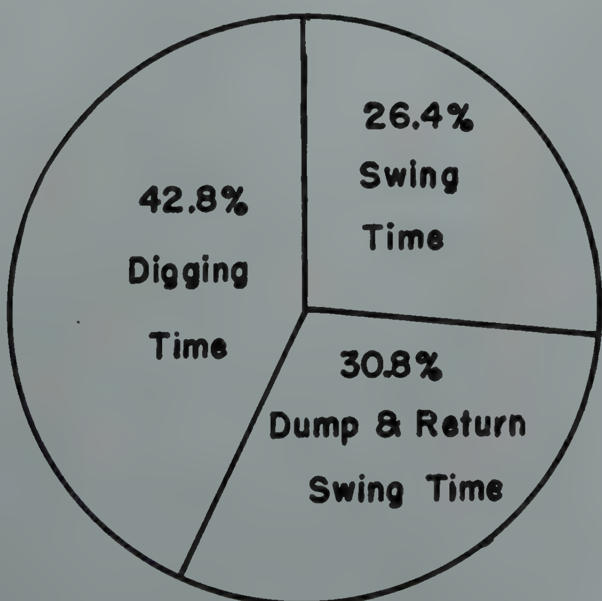

FIG 4—Time distribution for elements of digging cycle for a short period of 4 hr.

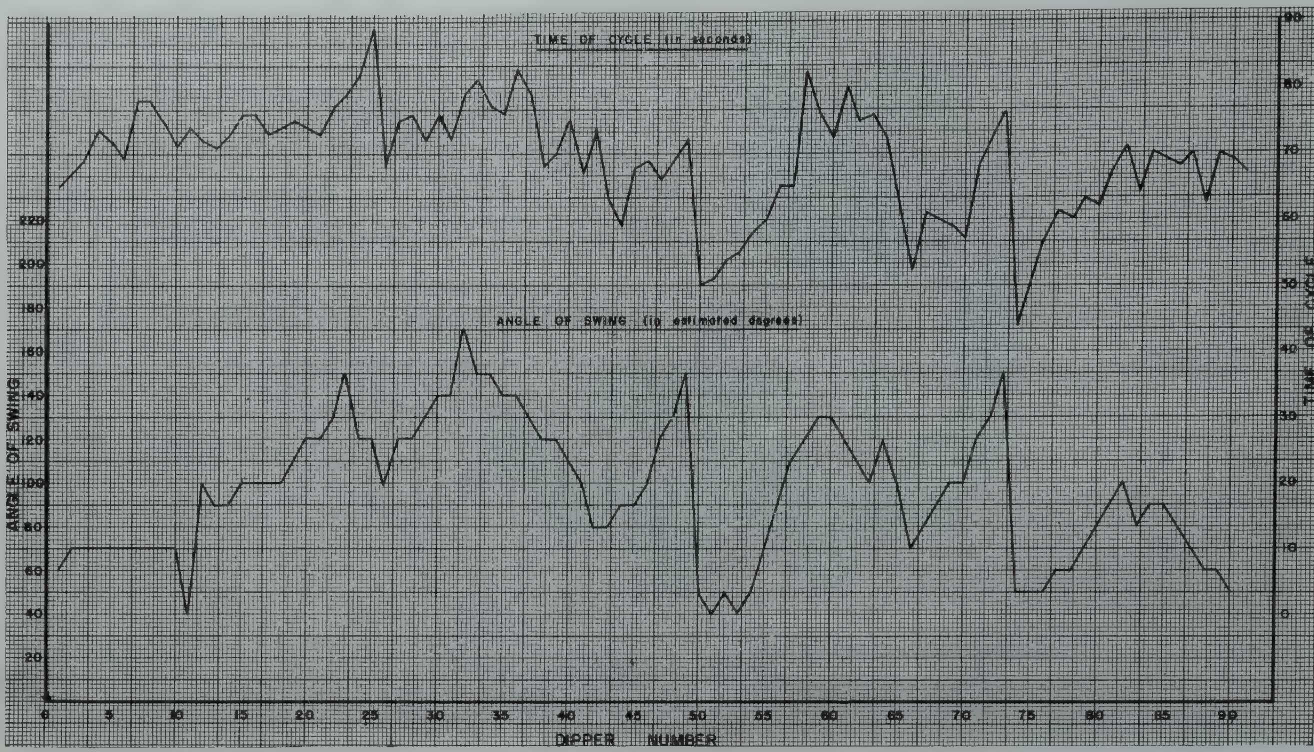

FIG 5—Relation between time of cycle and angle of swing.

dipper in hard or soft ground.

During 1947, the shovel consumed 3,267,800 kw-hr of electrical power. This consumption was segregated into averages of:

0.409 kw-hr per cu yd of overburden moved.

12.76 kw-hr per swing.

585 kw-hr per actual digging hour. During this same period the shovel made 256,087 swings, averaging 31.2 cu yd per swing in 1.31 min per cycle. This average load obtained with a 32 cu yd dipper represents a utilization of operational capacity of 97.5 pct.

DELAYS

The total delay time may be broken down as shown in Fig 3. (See also Table 2*C*.) The greatest loss of time was caused by mechanical failure and the second largest to miscellaneous delays such as moving power cable, oiling, blasting, and others. Power failures and electrical failures were the result of such causes as electrical storms, ice and sleet, and other power transmission difficulties. Reduction in the number of mechanical failures will come about as a result of improved design and manufacture of machine parts, and increased experience of the operators.

Whether the total delay time of 26.5 pct is excessive for this type of equipment could only be determined after an extended study of several shovels of the same type.

An examination of the data given above indicates that a further investigation of the following would be beneficial:

1. Study of maintenance procedures to reduce machine breakdown.

2. Studies of the design of machine parts to prevent breakdown.

3. Study of causes of power failure and their possible elimination.

4. Causes of miscellaneous delays and their reduction.

Acknowledgment

The Marion 5560 shovel described in this paper is located at the property of the Northern Illinois Coal Corporation at Wilmington, Illinois. The data summarized in Table 3 were recorded by Mr. Reiter, one of the authors. Other figures were supplied through the courtesy of the Northern Illinois Coal Corporation.

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The Flotation of Copper Silicate from Silica

By R. W. LUDT* and C. C. DeWITT,* Member AIME

THE use of froth flotation for the separation of minerals has become one of the most important of ore dressing processes. Its particular adaptability to the enrichment of low grade ores has made the process an important factor in the national economy. The methods have been extended to the recovery of a great number of minerals. Among the few minerals which have resisted efforts toward industrial flotation is chrysocolla, a hydrated partly colloidal copper silicate.

Chrysocolla, being a product of natural oxidation, has been found to occur in small quantities with many ores which are recovered by flotation methods. In present practice, these small quantities of copper silicate pass off with the tailings and are lost. The advantages to be gained by a satisfactory process for the recovery of chrysocolla is apparent. Any application of principles which points a way toward the satisfactory industrial flotation process for copper silicate would be of advantage. This paper presents an attack on this problem.

Two methods for the recovery of chrysocolla have been developed by the United States Bureau of Mines.[1,2] They have been successful on a laboratory scale but have been seriously restricted in industrial application by critical requirements in the procedure.

In one of the Bureau of Mines methods,[1] the ore is activated with sodium or hydrogen sulphide in an aqueous solution at a pH of 4. Amyl xanthate is then used as a collector with pine oil as a frother in the flotation process. An excess of sulphide acts as a depressant and the state of optimum conditions is difficult to control industrially.

In the second Bureau of Mines method,[2] soap is used as the collector at a pH of between 8 and 9. The difficulties with this process are that soap is not a specific collector, that heavy metal or alkaline earth ions cause the formation of insoluble soaps, and that a more acid solution causes the formation of a free acid which does not act as a collector for chrysocolla.

The problem of recovering chrysocolla by flotation involves the selection of a suitable collector. The collector molecule must be composed of an active polar group that has an attraction for chrysocolla, and of a hydrocarbon chain.

Certain dyes have been shown to have an attraction for certain minerals. Suida[3] found that hydrated silicates are colored by basic dyes. Dittler[4] showed that chrysocolla, among other colloidal minerals of acid reaction, preferentially takes up such basic dyes as fuchsin B, methylene blue, and methyl green. Endell[5] gave information to show that the colloidal material in clay may be determined by its selective adsorption of fuchsin.

A simple experiment, likewise, illustrates the difference in the adsorptive power of chrysocolla and of silica for the basic triphenyl methane dyes. When a mixture of chrysocolla and silica is immersed in a very dilute dye solution, less than 5 ppm, the chrysocolla is rapidly dyed and the silica is dyed more slowly. The difference is substantial but one of degree. Dean[2] showed that the dyes, crystal violet and toluidine blue, are taken up by quartz in an adsorption type process. The difference in the adsorptive power, however, offers the means by which a new collector may act.

San Francisco Meeting, February 1949.

TP 2535 B. This paper is based on part of a thesis submitted in partial fulfillment of the requirements for the degree of Doctor of Philosophy in Chemical Engineering.

Discussion of this paper (2 copies) may be sent to *Transactions* AIME before April 30, 1949.

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[1] References are at the end of the paper.

To form such a collector, a hydrocarbon chain must be attached to the dye molecule. This involves a process of organic synthesis.

Butyl, hexyl, and octyl hydrocarbon chains were selected for substitution in the malachite green molecule. For the purpose of identification, the alkyl-substituted dyes formed are called: butyl-malachite green; hexyl- malachite green; and octyl-malachite green. An outline of the procedure for their synthesis is given in the appendix.

It is generally recognized in the preparation of this type of dye that the chemical structure of some of the dye molecules varies. However, a uniform formula is attributed to the dye. Such a procedure has been followed in specifying the structure of these alkyl-substituted malachite green dyes. The structure is given on the basis of their properties as an homologous series of dyes, on their method of preparation, and on the purity of intermediates used.

Structure of substituted alkyl malachite green is:

$$\begin{array}{ll} & C_6H_4 \cdot N(CH_3)_2 \\ p\text{-}R \cdot C_6H_4 \cdot & CH \\ & \quad \cdot C_6H_4 \cdot N(CH)_2 \\ & \qquad\quad Cl \end{array}$$

Procedure

The flotation cell is a Bureau of Mines 100-g, batch unit provided with an air inlet at the bottom above which is a variable speed agitator. The agi-

tator and shaft are made of steel, the cell itself of lucite.

Froth discharges into a Buchner funnel where the concentrate is collected on filter paper. The liquid is drawn through the filter into a glass bottle by vacuum and is returned to the cell.

When the cell is charged, the total volume of pulp is made up of 100 g of solids and about 225 g of water. The exact amount of water in the cell depends upon the froth depth.

Flotation tests were run on synthetic ore mixtures of silica and chrysocolla. The minerals were ground separately in a wet charged ball mill, deslimed, dried, and screened. Only that material under 200 mesh was used in the tests.

Clean Ottowa sand was used as the source of silica. The chrysocolla was purchased from Ward's Natural Scientific Establishment, Rochester, N. Y.; this mineral was examined and identified by a minerologist. Dyed and undyed samples were examined under the microscope for colloidal and crystalline structure. Impurities of both crystalline and noncrystalline structure were found. If some of the impurities were crystalline copper-bearing minerals, they would not have been colored by the dye.[6] The sample of chrysocolla contained 23.5 pct copper.

Pine oil, as needed, was used as a frothing agent. The collector was added as an alcohol solution. The concentrate was recovered, dried, weighed and tested for copper by the method of Park.[7]

In some of the tests, the collector was added throughout the test in stage addition. The concentrate was continuously removed in the later runs, an induction period of 3½ min was allowed after each addition of collector. The induction period was included in the reported total flotation time.

Discussion of Results

In addition to the results shown in the report, many auxiliary runs were made to determine the general flotation procedure. Some of the conclusions from these preliminary tests are given below.

Particle size of the ore should be less than 200 mesh. This is well within the common particle size range in American industrial practice as summarized by Petersen.[8]

The presence of large amounts of slime, clay, or colloidal material complicates the recovery procedure. All of these materials should be removed before the flotation of chrysocolla.

The amount of air admitted to the cell influences the percentage of copper obtained in the concentrate. A large amount of air causes sufficient agitation to materially increase the amount of sand carried over in the concentrate. The quantity of air used in the runs was not measured and can be expected to give rise to some variation in results.

The use of common depressants such as tri-sodium phosphate, sodium silicate, acetic acid, hydrochloric acid, hydrofluoric acid, sodium hydroxide, and sodium carbonate show either no material improvement or a detrimental effect. All acid solutions that were used depress the chrysocolla. The use of sodium carbonate indicates some possibility for improved recovery.

The main purpose of the flotation tests is to show that appreciable recovery of chrysocolla from silica can be made by the use of an alkyl-substituted triphenyl methane dye when used as a collector. Optimum conditions of operation are not established but the influence of certain factors are noted. Operating conditions are found to have a great effect on the results. Check results vary to a great extent unless exact conditions are maintained.

The relative effectiveness of the three homologues of alkyl-substituted malachite green as a collector for chrysocolla is shown by a comparison of Runs 32, 31, and 34, in Table 1. The octyl malachite green is much superior to the compounds with the shorter hydrocarbon chains. All of the results shown in Table 1 are inferior to others shown in the report but they serve as a comparison for the three collectors Only the octyl dye indicates possibilities as an industrial collector for chrysocolla.

As in the use of many collectors, the time required for the collector to combine with the ore must be taken into consideration. In stage addition, the collector is added as flotation proceeds in order to prevent an excess of collector at one time. In other cases all the collector is added at one time and it is given an opportunity to combine with the ore in an induction period.

The data of Table 1 illustrate results under stage addition. A comparison of Runs 34 and 36 shows the effect of rate. The faster rate of addition gives a greater percentage recovery at a higher copper concentration. The faster rate of addition does, however, require more collector.

The use of a conditioning period further increases the enrichment of the concentrate. A consideration of the value called successive enrichment is a guide to the performance. Successive enrichment is an approximate value expressing the ratio of percentage copper in the concentrate to that left in the pulp. The values for successive enrichment on Runs 36 and 12A, show that a much richer concentrate is obtained by the use of an induction period.

The amount of collector required per unit weight of chrysocolla is not specifically determined but an indication may be obtained from Runs 12A

Table 1 . . . Flotation results

No conditioning period; pH, 7.7 to 8.0

Experiment Number	Total Collector		Heads, Per Cent Copper	Total Time, Minutes	Successive Concentrate			Total Concentrate, Per Cent	
	Dye	Ton			Copper, Per Cent	Recovery, Per Cent	Enrichment	Copper	Recovery
32-a	Butyl	0.01	0.235	2	0.74	6.9	3.3	0.74	6.9
-b		0.02		4	0.65	4.2	3.0	0.70	11.1
-c		0.03		6	0.41	3.2	1.9	0.60	14.3
-d		0.04		8	0.51	2.7	2.4	0.59	17.0
-e		0.05		10	0.37	2.6	1.8	0.54	19.6
31-a	Hexyl	0.01	0.235	2	0.99	10.6	4.6	0.99	10.6
-b		0.02		4	1.18	12.4	6.2	1.08	23.0
-c		0.03		6	0.94	18.1	6.2	1.01	41.1
-d		0.04		8	0.33	8.4	2.3	0.75	49.5
-e		0.05		10	0.31	9.8	2.5	0.61	59.3
-f		0.06		12	0.06	1.6		0.50	60.9
34-a	Octyl	0.01	0.235	2	1.27	6.5	5.7	1.27	6.5
-b		0.02		4	1.39	12.1	7.1	1.35	18.6
-c		0.03		6	1.58	9.8	9.0	1.43	28.4
-d		0.04		8	1.46	9.4	9.4	1.43	37.8
-e		0.05		10	1.55	10.7	11.8	1.47	48.5
-f		0.06		12	1.44	4.5	11.9	1.46	53.0
36-a	Octyl	0.03	0.235	2	2.26	17.4	11.4	2.26	17.4
-b		0.06		4	2.64	17.4	16.4	2.4	34.8
-c		0.09		7	2.12	17.4	18.1	2.2	52.2
-d		0.12		10	1.27	11.8	14.1	2.0	64.0
-e		0.14		12	0.93	10.0	13.9	1.7	74.0
-f		0.17		14	0.37	2.5	6.1	1.54	76.5

and 18A of Table 2. With the richer head material containing 1.275 pct copper, 0.002 g of octyl malachite green gives a concentrate containing 0.525 g of copper (44.8 pct recovery). With a head material containing 0.235 pct copper and the same amount of collector, it gives a concentrate containing 0.161 g of copper (68.4 pct recovery). On the basis of the copper recovered, the leaner head material requires greater amounts of collector.

When an excess of collector is added, the solution remains colored and the sand is distinctly dyed. Both excess of collector and increased time in a weaker dye solution cause the flotation of sand. The regulation and restriction of the amount of free collector in the cell have an important influence on the recovery.

Comparing Runs 18A, 14A, and 21A in which the head materials contain 1.275 pct, 0.47 pct, and 0.235 pct copper, respectively, both successive enrichment and percentage recovery are seen to increase with the leaner ore.

Conclusions

1. These tests have illustrated the application of physical and chemical principles to the flotation process. The results on synthetic mixtures are not to be construed as equivalent to data on industrial mixtures.
2. Alkyl-substituted malachite green dyes act as collectors for chrysocolla in synthetic silica-chrysocolla mixture. The octyl-substituted malachite green is more effective than the lower substituted homologues.
3. The amount of collector used and the method of adding it have an important influence on the extent of recovery. Better results are obtained when a conditioning period is used. Better percentage recovery and enrichment are obtained with leaner mixtures. Optimum conditions of procedure have not been thoroughly studied.
4. Clay or material of a colloidal structure interferes with the separation.
5. The results suggest that the industrial separation of chrysocolla from a silica gangue may with modification be developed along these lines.

Appendix

A series of three alkyl-substituted malachite green dyes was synthesized. Butyl, hexyl, and octyl-hydrocarbon chains were substituted in the malachite green molecule. The structure of these compounds has been assumed on the basis of their properties as dyes, on their method of preparation, and on the intermediates used.

For the purpose of identification, these alkyl-substituted dyes are called: butyl-malachite green; hexyl-malachite green; and octyl-malachite green.

Recognized methods of preparation were used with such deviations as were necessary because of the nature of the intermediates and products. The high molecular weight of the compounds, with consequent high boiling points, made purification difficult. In many cases, a distillation pressure of 5 mm was not low enough to prevent serious decomposition. These difficulties were overcome but they frequently resulted in low yields. An outline of the general procedure is given.

Alkyl chlorides were formed by the method of Helferich and Schaefer[6] by the reaction of the normal alkyl acid with thionyl chloride.

The alkyl chlorides were then reacted with benzene in a Friedels-Craft reaction to form a ketone. The general procedure of Ju, Shen, and Wood[9] was used, but benzene was used as a diluent.

The alkyl phenyl ketone was reduced to the alkyl benzene by a Clemmensen reduction. A method developed by Johnson and Kohmann[10] and described by Martin[11] was used.

The p-alkyl benzaldehyde was formed from the alkyl benzene by Bouveault's method.[12] The procedure is outlined by Weygand.[13]

The leuco-base of the substituted malachite green dye was formed in a regular dye reaction with dimethyl aniline. It was oxidized to the dye with lead peroxide. The general procedure is outlined by Cain and Thorpe.[14] After purification, the products were found to be solids with the general characteristics and color of the parent dye. Overall yields on the basis of the alkyl benzene used, varied from 5 to 15 pct.

Table 2 . . . Flotation Results

Conditioning period, 3.5 min; pH, 7.7 to 8.0; collector, octyl malachite green

Run Number	Total Dye, Ton	Heads, Per Cent Copper	Total Time, Minutes	Successive Concentrate			Total Concentrate, Per Cent	
				Copper, Per Cent	Recovery, Per Cent	Enrichment	Copper	Recovery
18A-a	0.04	1.275	4.0	10.3	23.6	11.1	10.3	23.6
-b			6.0	9.6	9.1	11.7	10.0	32.7
-c			8.0	7.4	7.7	10.0	9.5	40.5
-d			10	6.6	4.2	9.6	9.2	44.8
-e	0.08		14	6.6	8.5	11.2	8.6	53.0
-f			16	5.7	5.5	11.0	8.25	58.6
21A-a	0.04	0.47	4.0	5.2	29.0	15.1	5.2	29.0
-b			8.0	5.3	9.3	17.7	5.25	38.3
-c	0.08		12	3.05	11.4	12.0	4.5	49.5
-d			14	2.66	14.2	14.5	3.9	63.5
14A-a	0.04	0.235	4	4.55	31.2	27.5	4.55	31.2
-b			6	3.9	13.8	29.5	4.35	45.1
-c	0.08		10	3.7	10.5	34.5	4.2	55.8
-d			12	1.3	7.7	14.9	3.4	63.5
-e			14	0.65	3.3	7.7	2.8	66.5
12A-a	0.04	0.235	4	3.75	34.7	24.0	3.75	34.7
-b			6	3.3	20.6	33.0	3.56	55.3
-c			8	2.2	5.1	22.7	3.4	60.5
-d			10	1.2	7.9	15.3	2.8	68.4

Acknowledgment

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Recent Trends in Asbestos Mining and Milling Practice

By MICHAEL J. MESSEL,* Member AIME

Of the various minerals that occur in fibrous form known as asbestos, chrysotile is the variety most in demand for commercial uses, and, last year, over 683,000 tons of the various grades were produced in Canada and United States, exclusive of African and Russian production, which figures are uncertain. Production has not been able to keep up with the increased demand, and an acute shortage exists. Canada, Russia, and Africa are still the most important producers. However, Russia consumes most of its production at home and exports very little. The United States consumes over 60 pct of the total production and produces only about 4 pct of the total from deposits in Vermont and small quantities from Arizona.

This paper will review mainly, some of the more recent developments concerned with the extraction and processing of this fibrous mineral for various industrial uses, such as textiles, insulation, building materials and brake linings.

Four of the more important factors that have influenced recent developments are:

1. The lack of discovering and developing new deposits of any important size to supply the increased demand.
2. Rapid postwar expansion of industrial uses, especially in asbestos cement products, together with increased manufacturing facilities. European countries are again back on the market with their demand.
3. The ability of manufacturers to develop their technique of blending fibers and obtaining more utility value out of each ton of fiber, together with the utilization of shorter grades of fiber to obtain equally as good products. In the past, many manufacturers were wasteful in their use of fibers.
4. The struggle for reduced operating costs, in face of increased wages and prices of supplies, together with the necessity, at some mines, to change the method of mining from open quarry to underground.

Most of the recent capital invested in the asbestos mining industry has gone into more efficient extraction of the fibers from the existing ore, especially in the recovery of the shorter grades. There has been very little new plant expansion. Progress in the utilization of shorter fibers has been so far reaching that some of the mining companies are retreating present tailings as they leave the mill, and others are considering retreating of the old tailing dumps for recovery of fibers that, until recently, were not saleable and were discarded as part of the tailing. Many of these dumps contain much valuable fiber from days when milling was very inefficient.

It might be appropriate to mention here that the present known asbestos reserves of proved commercial value (excluding Russia) are being depleted at an alarmingly rapid rate, about 10,000,000 tons of ore is being mined annually. No new deposits have come into production during the last 15 years, with the exception of certain deposits in Africa, which are the most promising, and minor developments in Canada, Venezuela, Cyprus, Australia, and Brazil. The African deposits are the only ones that hold promise of developing large reserves. Unless new deposits are developed, in twenty years or maybe sooner, the supply picture will not be a very bright one. Stimulated by the acute shortage, increasing fiber prices and technical developments, many new areas are being prospected in Canada, United States, New Zealand, Newfoundland, South America, Europe, Africa, and China.

From information now available, it appears that no real progress has been made in duplicating this rare development of nature by producing synthetic asbestos. The most promising experiments were carried out at the University of Leipzig, but they seem far from practical. There has been marked progress in fiber glass manufacture, but so far, it has replaced asbestos only in very few instances for minor uses.

Asbestos ore and the fiber it yields are different in some respect at each mine and vary usually in the following:

1. Percentage of fiber content by weight to the gangue.
2. Variations in proportions of long and short fibers, some deposits being predominantly long fiber and others short fiber.
3. The hardness of the rock in which the fiber occurs.
4. The nature of the fiber, as to

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being harsh or crudy, difficult to open and fluff up, or easy to open. Asbestos fiber has never been opened to its maximum and each fiber bundle can still be further reduced into smaller fiber bundles. However, fibers are opened to a certain degree for various manufacturing purposes, and this opening should not go beyond such mechanical treatment as will result in the breakdown of fiber length.

5. The tensile strength of the fiber.

With all these variables involved in each deposit, and usually within the same deposit, the various developments at different mines have varied to some degree.

FIG 1—Milled asbestos fiber.

Review of Some of the More Important Developments

PROSPECTING AND LOCATING DEPOSITS

Discovery of most of the present important asbestos deposits has been purely accidental, and, in most instances, were made by farmers or wood cutters. By strange coincidence, these original discoveries of about 60 years ago are the most valuable and extensive deposits, most of which are still being worked today.

Geophysical methods have been used only with partial success. While certain electrical methods can outline fairly closely the extent of serpentine formations, which are favorable to the occurrence of asbestos, they give no indication of the presence of fiber or of its quantity. However, geophysical prospecting did show up extensive shearing in one deposit, a condition which is often favorable to occurrence of asbestos.

More careful methods of systematic core drilling and sampling are now used to get a more accurate appraisal of the deposit. However, samples of the fiber still must be obtained and tested in the laboratory and by actual manufacturing methods before it is certain that the fiber possesses the necessary characteristics to be of commercial value.

MINING METHODS

The conventional methods of mining are those ordinarily used in metal and nonmetal mines. In most cases, the deposits originally occurred as surface outcrops and open quarry methods were applied. In Canada, as the economic limits of quarrying were reached, the underground method of block caving was applied successfully and economically. This method has been described previously,[1] and some of the outstanding features of its application are as follows:

1. It can be successfully used in a slippery and highly sheared rock such as serpentine.
2. Concrete and steel should be used in place of wood for all underground supports and chutes.
3. At one mine, diamond drills have been used recently for undercutting the block, by drilling long holes and blasting, thus reducing the necessity of drifting.
4. Another company is considering the use of scrapers in the grizzly drifts, thus doing away with considerable expensive grizzly and chute work.

In Africa at one mine, a combination of shrinkage stoping and top slicing is applied to permit sorting of the ore in the stopes.

In quarrying, the current trend is to abandon drilling in a series of small benches, and to work benches up to 150 ft in height by means of blasthole drills, drilling holes 6⅝ in. diam spaced on 18 ft centers. At some mines, air drills mounted on derricks are used. At one location these drills are working a 90 ft face, drilling 70 ft holes with a 3½ in. starting bit, and a 2½ in. finishing bit. Supplemented with a row of holes at the base, large blasts are brought down successfully. With these high benches, blasts of 100,000 tons are not uncommon, and while some operators claim this has a tendency to destroy fiber length, it is probable that the economic advantages more than compensate for any such possible reduction.

[1] References are at the end of the paper.

For loading the ore, mechanical crawler type shovels of various sizes are used, the emphasis being on units which have a fast swing and easy maneuverability.

Haulage methods still vary considerably from one mine to another, depending on conditions. At one time, railways were used in nearly all quarries; the tendency now is towards truck haulage, with such units as Euclid trucks and Lynn tractors. At one mine, the use of belt conveyors and shuttle cars has been seriously considered.

CRUSHING, DRYING AND MILLING

Because of the rather low ratio of asbestos fiber to rock, an average of 5 to 6 pct, large tonnages of ore have to be milled in order to operate on a commercially successful basis. A capacity of 1000 tons per day is considered very small and usually the minimum unit.

Basically, there has been no change in the asbestos milling process, which still consists of primary crushing in two stages to reduce the ore to minus 1½ to 2 in., followed by drying the ore for removal of moisture to permit proper milling. The milling process consists of further crushing to release the fiber from the rock along cleavage planes, screening and aspirating the free fiber, followed by further fiberizing in specially designed machines which fluff up the fiber, and further screening and aspirating by air suction of the free fiber into cyclone separators. (Fig 1.)

This process is repeated in two or three stages, until all marketable fiber is removed and the remaining rock and fine sand is discarded as tailings. The collected fiber is then passed over screens for cleaning and removal of fine sand and unopened spicules of

fiber and rock particles. The clean fiber is then classified and graded according to length. There are about 25 grades depending on the proportion of the various length of fibers.

Fiber is still valued and sold according to length as determined on a standard Quebec testing machine, though some mines have their own standard and do not follow the Canadian practice. While preservation of length is still important in milling, more emphasis is now being given to efficient extraction of every marketable pound of fiber. Demand has carried this to the point where some of the shorter grades, known as asbestos sand, now contain up to 50 pct, or more, of granular rock particles. This grade is blended with other short grades for the manufacture of asbestos floor tile.

Drying

For drying the ore, (which contains from 1 up to 20 pct moisture), no new types of dryers have been developed. Either the vertical stack type or the rotary dryer is used. More attention is being given at present to insulation and to more efficient use of fuel. The relative efficiency of the drier is now about 50 pct, whereas not long ago 30 pct was considered good. For efficient milling the rock must be dried to 1 pct moisture or less. The dried rock is then stored in large silos or bins. These silos are usually constructed of tailing material which forms natural banks with sloping sides in the form of a hopper that is covered with a structure at the top of which is located a feed conveyor. At the bottom, there is a concrete tunnel with draw points. Silos up to 60,000 tons in capacity have been constructed in this manner at a cost of around $5 to $6 per ton of storage. Such silos are economical in construction and permit large storage of dry ore for more effective curing, and also for blending of the ore to attain a more even and constant mill feed. The use of surge piles of ore at various points in the process, from quarry to mill, makes operation more efficient and has been very successfully applied at one mine. These piles require no covering and material is drawn out below through a concrete tunnel equipped with draw points.

Milling

Milling is a dry process. In each ton of ore the fiber length may range from 1½ to 1⁄16 in., so no fixed flowsheet is possible, as the flowsheet must be adjustable to meet varying conditions. Certain grades of fiber demand special equipment.

The four factors that can be partially controlled in milling are:

1. The efficiency of the extraction.
2. Maintaining the length of fiber as it originally comes from the deposit.
3. The degree of opening or fluffing of the fibers.
4. The amount of "drops" (−35 mesh) and granular dust remaining in the finished fiber.

Fine Crushing and Fiberizing

The use of Symons Shorthead cone crushers set to as fine as ⅛ in. is finding a place in nearly every flowsheet. The advantage of this equipment is that it frees the fiber from the rock by crushing it along the plane of weakness without destroying the fiber length. It also produces a harsh primary fiber which is practical for some uses. The crushing also prepares the rock particles for the operation following screening and aspirating, that is, for fiberizing or fluffing of the fiber, together with freeing more fiber and making it amenable to air suction.

Three types of machines have been developed for freeing fiber from rock, and while somewhat similar in principle they may give widely different results on the same ore. The principle of these machines is that of a hammer mill, which by means of rotating hammers, strikes or throws the particles against corrugated plates or jaws, the impact and attrition causing the fibers to open and fluff.

These machines are the Jumbo, Torrey Cyclone, and Impact Mill.

The Jumbo consists of a horizontal steel shell, 36 in. diam by 72 in. long, lined with corrugated liners, in which is mounted a shaft that has steel arms with beater tips attached. This shaft rotates at 600 rpm. The material enters at the top on one end and is discharged at the bottom of the other. This Jumbo, while efficient, has a tendency to cut up and grind the fiber. The present trend is now to replace it with a combination of crushers and vertical type mills.

The Torrey Cyclone and Impact Mill consist of a steel shell, approximately 4 to 5 ft diam by 5 ft high. The shell is lined with corrugated manganese plates or jaws. In the center is a rotating shaft to which the beater arms are attached. This shaft rotates at 900 to 1000 rpm.

The Torrey has two sets of hammers separated by an intermediate hopper. The material entering at the top is thrown by the centrifugal force of the hammers against the jaws. The material is then guided toward the center and the action is repeated by the next set of hammers.

The fiberizing in this instance is done by impact against steel jaws, though there must be some disintegrating action by inertia. The fundamental difference between the Torrey and the Impact Mill, is that in the Impact the intermediate hopper is eliminated; the material falls freely by gravity and is struck by the revolving hammers. Disintegration takes place predominantly by simple inertia.

Neither of these machines is ideal, both have a tendency to cut up some fiber and create rock dust. Mechanically, the Impact is superior but requires more power per ton of material fiberized. A more efficient machine may be a modified combination of the two, and this is now being developed.

Screening

Screening is very important in asbestos milling and it occupies about 60 pct of the total floor space in a mill. It is important to recognize that the fundamental functions of the various screens change from one end of the flow to the other.

At the beginning, in the rock flow, the primary function is some separation and removal of sand, but fundamentally it is to form an even layer of ore so that the fluffed up fiber particles may be aspirated by air.

For treating the collected fiber, the screen function is removal of dust and fine sand, and formation of a bed to facilitate aspirating the fiber by selective air so that unopened particles of fiber and rock will not be lifted but returned to the primary rock flow or special circuit for further fiberizing.

The function of screening of cleaned fiber is to separate the various lengths of fiber.

Two main types of screens are used. The reciprocating type actuated by an eccentric is known as a shaking screen, the usual size is 5 by 10 ft, and speeds range from 200 to 450 rpm. The other type is one with a gyratory motion, actuated by a vertical eccentric head. At one time, only shaking screens were used in the industry, and many mills still retain the practice, but some mills have gone exclusively to the gyratory type. There have been very

marked improvements mechanically on both types of screens.

One type of gyratory screen is claimed to have about twice the capacity as compared to the shaking screens, but this is in some instances only, depending on the objective. However, there is little doubt that the gyratory screens are considerably more efficient in the fiber cleaning and classification circuits.

Fiber Collectors and Air Systems

Asbestos mills are still a maze of air pipes, cyclone collectors and fans, which are used for aspirating and collecting the fiber. The principle of separating fiber from rock depends on a fan-created vacuum in the collectors. The resultant air suction lifts the fiber off the screens through the hoods and piping to the collector. About 1 to 2 in. vacuum is required on the rock screens and ½ to 1 in. on the fiber screens. Only developments of a minor nature have taken place here, such as, more efficient collectors, improved piping layouts, and more efficient fan blades. About 5000 cfm is required per screen, and air is such an important item in the process that it takes about 20 to 25 pct of the total horsepower used in the mill operation.

Grading and Classification of Fibers

After the fiber is cleaned, it is usually already partly classified and the final adjustments for standard test are made by using screens, or, more frequently, by the rotary type graders. The different mines have developed various types and sizes of their own. Basically, this unit consists of an enclosed trommel of perforated plate which may or may not rotate, and has a central shaft with beater arms which are inclined in the direction of the flow. The materials enter at the top of one end and are forced across to the other end by the beaters. The shorter fibers are forced out through the screen openings, and longer ones are carried through and discharged at the end. Various size openings may be used in the same trommel to give different size products. A standard machine now frequently used is 28 in. diam by 10 ft long, and rotates anywhere from 200 to 500 rpm. The capacity of these units is usually 1 to 2 tons per hour for long fibers and 4 to 5 tons on the shorter grades. The resulting products from these graders are blended to obtain the desired test and specifications.

Dust Collecting

In asbestos milling, which is a dry process, considerable dust is created in the various operations, together with a large amount contained in the air from the fan exhausts, as the fiber collector is really only a partially efficient separator. Some shorter fibers escape into the fan, and these, if recovered, have commercial value.

In the past, this air has usually been sent through ducts to a large settling chamber, where the heavier material is dropped because of reduced velocity, and returned to a special circuit or department in the mill flow. At the end of the dust building, there is a chimney for the exhaust dust. In one instance where this dust has been objectionable, one company has installed a Cotrell system.

However, a recent successful development has been the application of bag type collectors with automatic shaking devices which usually can be made to fit very nicely into the top of the conventional mill building. These units not only recover the valuable fiber, but also the dust, so there is a very small amount of exhaust dust in the atmosphere. Some companies are experimenting with the possibilities of recirculating this air. About 98 pct efficiency by weight is claimed for these units, and at present, there are many installations under way. The cost is approximately 20 cents per cubic foot of air, and about 1 sq ft of filter cloth is necessary for filtering 3 cu ft of air.

Tailings Treatment

Mill efficiency is usually based on fiber recovery, as compared to weight of the rock. This is not based on the original fiber present in the ore, as there is no accurate way to determine this, but on the pounds of marketable fiber that can be recovered from a ton of ore. Therefore, the term "fiber content" varies with conditions, and in the past, due in part to inefficient milling and the non-marketable shorter fibers, many tons of now valuable fiber reached the dumps.

Due to the demand for shorter fibers, some mills have installed Whizzer type air separators or other modified types of air separators on their tailings, rather than adjust or modify the existing flowsheet.

At one mine, a plant is now being constructed to retreat the old tailings dumps by a wet method. The tailings, as they come from the dump, are screened and the plus ⅜ in. is discarded as of no value. The minus material is sent to cone type classifiers, where the fiber is floated off in water and sent to filters. Later, it is removed, dried and fluffed up, and a clean commercial fiber results.

Bagging of Fiber

Asbestos fiber, when processed, becomes bulky and difficult to handle. The various grades of fiber are packed either by hand or machine, but more usually by plunger type bagging machines, in 100 pound lots, into either jute or paper bags.

The bagging machines, used at present by the various companies, are rather cumbersome, and there is certainly plenty of room for improvement here. Bags alone, today, cost about $4 to $6 per ton of fiber.

If some method of compressing and bailing, or bulk shipment, could be worked out, it would mean considerable saving to the industry.

Summary

Development of the present method of extracting and milling asbestos fibers has taken many years of practice, and the ingenuity of many capable men, to achieve the refinements and controls available today in producing fibers for manufacturing requirements to standard specifications.

In the asbestos milling and manufacturing industry, there are so many variable factors involved, that it is easier to claim progress than to prove it. However, the trend is toward more efficient equipment and more efficient utilization of the fibers. In general, there is a very cooperative spirit between the various companies which should result in very marked progress for the future.

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A Technical Study of Coal Drying

By G. A. VISSAC,* Member AIME

Moisture in Coal

MOISTURE in coal must be considered as an impurity, just the same as ash, from the standpoint of utilization of the coal. Being incombustible, it reduces directly the heating value of the coal, and in addition absorbs heat for its evaporation. Its presence means useless expenditures in handling and transportation. In coke plants, extra moisture reduces capacity and may cause damage to brick work and equipment.

Accordingly, the removal of extra moisture can be considered just as important as the removal of other impurities, such as ashes, in the modern coal preparation plant.

Moisture, which can be removed by heating the coal up to a temperature of 100°C, may be retained in various forms:

1. As a film, on the surface of each coal particle, and in the interstices between particles, retained by capillary forces.
2. Or "occluded" inside the coal particles. This occluded moisture may be either free moisture (as in a sponge), or hygroscopic moisture which varies with atmospheric conditions, (also called "regain").

These latter forms of moisture are particularly common in "young" coals (subbituminous and lignites); bloom coals (seam outcrops); fusain; and carbonized products.

In our study of coal drying, we shall consider only the removal of free moisture, exclusive from hygroscopic moisture.

Dewatering

If we reserve the name of drying to the removal of water by evaporation, we must consider the initial phase of the mechanical removal of free moisture as a distinct operation covered by the term dewatering.

In all cases the free water carried over the surface of the coal particles or in their interstices, or in their pores, is retained by capillary forces. Dewatering is accomplished by breaking or counteracting these capillary forces; removal of as much water as possible by dewatering methods is usually advisable, as the cost of these operations is generally much less than by evaporation.

The most common methods of mechanical dewatering are:

1. "Pressure piling," which reduces the interstitial spaces, accomplished in dewatering bins or over dewatering screens.
2. Or dynamic methods, such as used in centrifuges or over vibrating screens.

We shall only mention the "preferential wetting" method, in which surface water can be displaced by hydrocarbons, as offering possibilities, but which, to our knowledge, has not reached yet a practical development.

But we must point out that the capillary forces retaining water on the coal surfaces, decrease considerably with increased temperatures. This is the principle used in all modern dishwashing machines; by using very hot water, dishes are extracted almost dry.

In line with this development, we favor the type of dryers including a dewatering section; as the coal enters the dryer and is gradually brought up to higher temperatures, its dewatering ability is increased and advantage can be taken of this conditioning, resulting in increased drying efficiencies and reductions in drying costs.

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Heat Drying

In the final phase, the remaining moisture must be evaporated. Coal and water must be brought up to the chosen temperature of evaporation, and heat must be supplied to fill the requirements of the latent heat of evaporation of the water to be removed.

Accordingly, drying becomes largely a problem of heat transfer, and drying methods can be classified accordingly, namely:

1. Radiant transfer.
2. Transfer by surface contact and conduction.
3. Transfer by hot gas contact.

The first method is not applicable to coal drying; the second method is used in the old type rotary dryer. The third method, the most commonly used in modern coal dryers, will be the only one considered here; and, of course, we shall deal with continuous types of dryers only.

The mechanism of complete drying is really very complex—several phases are involved:

1. The constant rate period.
2. The uniform falling rate period.
3. The varying falling rate period.

As most of our practical coal drying problems deal with wet coals (over 6 pct of moisture), and do not require complete drying (under 1.5 pct), we shall deal with the first condition only, namely the *constant rate drying*.

Dryer Calculations

Instead of presenting the algebraic formulas, we believe a concrete example will provide a clearer illustration.

Assume a feed of wet coal at the rate

of 60 tons per hour with a free moisture content of 12 pct; it is required to bring the free moisture content down to 2 pct.

We assume our dryer will remove mechanically 5 pct of moisture, leaving 5 pct to be removed by heat drying.

VOLUME OF GASES REQUIRED

(Summer conditions 60°F.)

A feed of 60 tons per hour is 1 ton, or 2000 lb per minute.

The moisture to remove is 5 pct of 2000, or 100 lb per minute (dry basis).

Assume evaporation at 120°F (temperature of gases at dryer's exhaust) and 60°F as outside temperature.

1 cfm of air carries:
at 120°F and at
full saturation 0.005 lb water per cfm
and at 60°F 0.001

a difference of 0.004

And if we assume a 75 pct evaporation,
1 cfm of hot gases will remove 0.003 lb of water

Accordingly, the air required on the basis of evaporation alone is: 100/0.003 or 33,333 cfm (measured at sea level, and 70°F).

TEMPERATURE REQUIRED FOR HOT GASES

If we call T this temperature,
$33{,}333 \times 0.073 \times 0.25 \times (T - 120)$
will give us the number of Btu's available, when 33,333 cfm of gases — density = 0.073, sp. heat = 0.25, cool off from T, down to 120°F (temperature of evaporation).

This amount of heat per minute must be sufficient to:

1. Evaporate 100 lb of water at 120°F
(latent heat = 1024),
or
$100 \times 1024 = 102{,}400$ Btu
2. Heat coal and water from 60° to 120°; namely:
Coal: $2000 \times 0.25 \times (120 - 60)$
Plus water: $100 \times 1 \times (120 - 60) = 36{,}000$ Btu
(sp. heat of coal = 0.25)
3. Take care of radiation losses:

On the basis of a dryer with 2000 sq ft of insulated surface and 2000 sq ft of exposed surface, we have
2000 sq ft at $0.2\ (T - 60)$
plus 2000 sq ft at $2.5\ (T - 60)$ per hour
(60 being the outside temperature)
or per minute: $90T - 5400$ Btu

The equation of heat balance is then:
$33{,}333 \times 0.073 \times 0.25 \times (T - 120)$
$= 102{,}400 + 36{,}000 + 90\ T - 5400$
or $T = 400°\text{F}$ (approximately).

TOTAL HEAT REQUIRED

As per the above, the heat required is total of (1), (2) and (3), namely 169,000 Btu's per minute, or roughly, 10,000,000 Btu's per hour.

WINTER CONDITIONS

(Outside temperature 0°F.)
Evaporation at 110°F.
Volume required:
Lb of water per cu ft at 110°F = 0.00376
Lb of water per cu ft at 0°F = 0.00006

0.00370

At 75 pct evaporation —
1 cu ft will remove 0.00278
and for 100 lb per minute we require
100/0.00278 = 36,000 cfm
(measured at sea level and 70°F).

Hot Gases Temperature

T given by equation:
$36{,}000 \times 0.073 \times \frac{1}{4}\ (T - 110)$
$= 100 \times 1020 \times 2000 \times \frac{1}{4} \times (110 - 0)$
$+ 100 \times 1 \times (120 - 0)$
$+ 2000 \times 0.2\ (T - 40) \times \frac{1}{60}$
$+ 2000 \times 2.5\ (T - 40) \times \frac{1}{60}$
or $657\ (T - 110) = 165{,}400 + 90\ T$
or $T = 418°\text{F}$.

Total Heat Required

200,000 Btu per minute
or 12,000,000 per hour.

FURNACE REQUIRED

Total heat required = 170,000 Btu's per minute.
or with coal at 12,600 Btu and at 75 pct combustion efficiency
170,000/0.75 × 12,600
or 18 lb of fuel per minute
or 1080 lb per hour
at 42 lb per sq ft.

The required grate area is 26 sq ft.

Combustion space at 50,000 Btu per cu ft per hour required
= 200 cu ft or 7.7 arch.*

DRYER FAN

Volume required = 33,333 cfm at 70° at sea level
or 33,333/0.858 = 40,000 cfm at 70° at 4000 ft.

* Combustion here at 300 pct excess air, giving: 15,000 cfm hot gases and besides 18,000 cfm of cooling air are added, giving the required total of 33,000 cfm at the dryer.

water gauge 4 in. at 70° at sea level
or 4/0.858 = 4.66 in. 70° at 4000 ft.

Power required:
$$\frac{(40/33)^3 \times 0.858 \times 0.000157 \times 33{,}333 \times 4}{0.65}$$
or 49.5 hp

BLOWER FAN AT FURNACE

Required 15,000 cfm (equal to volume hot gas in furnace)
Water gauge = 1 in.

COLD AIR DUCTS

Required 18,000 cfm of cold air (see furnace calculations).

HOT GASES DUCTS

Volume at 70°F sea level is 33,333 cfm at 400°F at 4000 ft elevation volume is:
33,333/0.858 × 0.616 = 63,000 cfm.
Desired speed about 2000 fpm.
Section required about 30 sq ft.
Proposed area 5 × 6 ft.

Influence of Hot Gas Temperature

The Table 1 illustrates the effect and consequences of a change in the hot gas temperatures.

In Table 1 we have condensed the operating data corresponding to two different hot gas temperatures. Increasing the hot gas temperature from 400° to 660°F, reduces the required volume from 34,000 cfm down to 18,800 cfm, and the fan consumption of power from 38.50 kw-hr down to 20.25 kw-hr, but the Btu's required are increased by 2,000,000 Btu per hour.

However, this is based on the assumption that proper arrangements are made to maintain the water gauge constant (in our case equal to 6.35 in.). But if we had to use the same size of ducts and the same coal bed resistance, in Operation No. 1, the water gauge would be increased by 1.5 and the required kilowatt hours by 2.25, to 86.60 kw-hr, an increase of 66.6 kw-hr over Operation No. 2.

Assuming a ten hour day operation, and taking this most unfavorable condition for Operation No. 1, the final balance will be:

Condition No. 1 uses 666 kw-hr more per day (10 hr), or at 0.005 cents per kw-hr, an excess cost of $3.33 per day, but uses one ton of coal less per day, which may be worth the same value.

Table 1 . . . Dryers Operating Data for Two Different Hot Gas Temperatures

	Operation 1	Operation 2
Temperature of evaporation	120°F	140°F
Volume of hot gases	34,000 cfm	18,800 cfm
Temperature of hot gases	400°F	660°F
British Thermal Units required:		
Evaporation (100 lb)	102,400 Btu	101,300 Btu
Heating water and coal	36,000 Btu	48,000 Btu
Radiation losses	30,600 Btu	54,000 Btu
Total per minute	169,000 Btu	203,000 Btu
Total per hour	10,140,000 Btu	12,198,000 Btu
Horsepower required for dryer fan	50 hp	27 hp
Kilowatt hour	38.50	20.25

Normally, with a dryer adapted to take care of the increased volume, that is, the same water gauge, the operating balance is definitely in favor of the low temperature, namely:

1. Operation No. 1, an increased power consumption of 18.50 kw-hr, or 185 in a 10 hr day, or 0.92 cents per day.

2. Operation No. 2, an increased coal consumption of 1 ton per day, worth between $3.00 and $3.50.

In order to give a wider picture of the influence of the hot gas temperatures, we have worked out in Table 2 the corresponding volumes and temperatures with temperatures of evaporation from 110° to 145°F.

Table 2 . . . Required to Evaporate One Pound of Moisture per Minute

V—Volumes of hot gases required, measured at 60°F and sea level.
T—Temperature of hot gases at duct in Fahrenheit, corresponding to various temperatures of evaporation.

Temperature of evaporation, degrees F	110	115	120	125	130	135	140	145
V, cfm	495	405	340	289	251	220	188	167
T, degrees F	300	350	400	460	520	580	660	750

Rate of Evaporation

QUANTITY

The rate of evaporation is given by the formula:

$$W = C \times V^{0.8} \times (e_w - e_a)$$

where:

W is pounds of water evaporated per square foot per hour.

V is velocity of hot gases in feet per minute.

e_w is vapor pressure in inches of mercury at water temperature.

e_a is vapor pressure in inches of mercury in atmosphere.

C is a constant characteristic.

The above formula illustrates again the influence of the factors involved. A higher temperature of evaporation means a higher value for e_w, but a lower volume of gases and a lower value of V.

TIME

If Q is the heat energy conducted in time t, through a material of sectional area A, normal to the flow of heat and of thickness D in the direction of the flow of heat, with a temperature difference T between its surfaces, Q is given by the formula:

$$Q = K \times \frac{ATt}{D}$$

K is a constant characteristic.

or

$$t = \frac{QD}{KAT}$$

In other words, the time necessary to the required heat exchanges is reduced by the use of higher temperatures, factor T, but this is partly compensated by the increased heat energy required, Q.

But the above formula clearly indicates the advantage of a large area, A, in reducing the time required to the heat exchanges.

Cooling and Regeneration

As the dry coal and the exhaust gases leave the dryer at a higher temperature than when they were introduced, we shall lose a certain amount of Btu's. In order to increase the dryer's efficiency we must endeavor to recuperate and regenerate the maximum possible amount of these Btu's carried away by the dry coal and the exhaust gases.

The best practical methods in our experience are as follows:

1. Blow some cold air over the coal coming out of the dryer

2. Spray water in the dryer's exhaust.

COOLING OFF THE COAL

In fact this cooling off will be used to evaporate a certain amount of the water remaining in the coal. If the coal is cooled off from 120°F, down to 70°F, the amount of Btu's liberated at the rate of 2000 lb of coal per minute are:

$$2000 \times 0.25 \times (120 - 70) = 25{,}000 \text{ Btu}$$

If we use this amount of heat to accomplish an extra amount of drying, if p is the extra weight of moisture evaporated per minute, and V the volume of cold air originally at 30°F and heated to 80°F, we have the equation:

$$V \times 0.08 \times (80 - 30) \times 0.25 + p \times 1000 = 25{,}000$$

Saturation at 80°F is 0.0013 lb per cu ft.
Saturation at 32°F is 0.0003 lb per cu ft.
a difference of 0.001 lb per cu ft.

We can admit that every cubic foot will evaporate 0.001 lb of moisture, then

$$0.001 \times V = p$$

Solving the above equations gives us:

$$V = 12{,}500 \text{ cu ft per minute}$$
$$p = 12.5 \text{ lb of water}$$

or, on the basis of 2000 lb of coal per minute, an additional drying of over half of one per cent, made available without additional heat.

The corresponding increased consumption of power will be about 5 kw per hour,
or, on a 10 hr shift, 50 kw-hr
or, 0.25 cents per day (at half a cent per kw-hr).

The saving in coal consumption at the furnace on the basis of a 50 pct efficiency will be
2 lb per minute
or 120 lb per hour
or 1200 lb per day.

REGENERATION

The cooling off by water of the exhaust gases from say 120°F down to 60°F will recover, with a volume of 35,000 cu ft at 0.06 lb per cu ft.

$$35{,}000 \times 0.06 \times 0.25 \times (120 - 60) = 31{,}500 \text{ Btu}$$

The circulating water of a washery operating in a closed circuit is used for this purpose; this regenerated heat is used to preheat the coal before it is fed to the dryer.

In addition to their technical value, cooling and regeneration offer many material advantages:

1. Cooling reduces the danger of spontaneous combustion, decreases sweating of loaded coal.

2. Regeneration assists in the heating of the preparation plant, the thawing of frozen coal, and its mechanical dewatering (by reducing surface tensions).

Analysis of Results
Measures of Efficiency

The efficiency of a heat dryer is equal to the quotient of the heat actually used for evaporation, by the total heat supplied.

Example: (refer to Table 1, Operation No. 1, hot gases at 400°F)
Heat used for evaporation
102,400 Btu per minute
Total heat required by dryer
169,000 Btu per minute
Efficiency: 60 pct (approximately)

Operation No. 2 (hot gases at 660°F)
Heat used for evaporation
101,300 Btu per minute
Total heat required by dryer
203,000 Btu per minute
Efficiency: 50 pct

This is another illustration of the advantage of operating at low temperatures.

The total efficiency of a drying plant is the quotient of the Btu's actually used for evaporation, by the Btu's actually supplied to the furnace, or

Operation No. 1
$0.60 \times 0.75 = 45.0$ pct

Operation No. 2
$0.50 \times 0.75 = 37.5$ pct

Cooling and regeneration will increase the above efficiencies as follows:

Operation No. 1: The theoretical heat required to evaporate
5 pct by the dryer
0.5 pct by cooling
would require for the system—
$102{,}400 + 10{,}240 = 112{,}640$ Btu's

As previously indicated, the total heat required is	169,000 Btu's
Less, recovered by regeneration	31,500 Btu's
or	137,500 Btu's

The actual dryer's efficiency is increased from 60 pct up to

112,640/137,500, or 82 pct

For a true comparison between the various drying devices, other factors must be taken into consideration.

Referring to our dryer calculations, we have estimated at 75 pct the saturation of the exhausted gases in the dryer. This is an actual minimum, obtained by experience, and which applies to most dryers. However, in actual practice this figure may have to be checked; a lower saturation will indicate lack of sufficient contact between coal and gases, because of a faulty design or a faulty operation.

From the above discussion it appears desirable to operate a coal dryer at low temperatures, in order to obtain not only the maximum efficiency, but also the best safety of operation; in this connection, it will be important to keep in mind the following kindling temperatures:

	Degrees F
Bituminous coal	600
Bituminous slack	500
Anthracite dust	570
Lignite dust	240
Anthracite slack	750
Lignite slack	435

(with slight variations between fields)

However, in order to operate at low temperatures and still handle large tonnages of coal, a coal dryer must be able to handle large volumes of gases, and special mechanical devices will be required to that effect.

Another frequent advantage of the operation of a dryer at low temperatures is in the possibility of using the exhaust gases available from existing boiler plants, or carbonization plants.

Generally speaking, all dryers can be adapted to dry fine coals; the mechanical problems involved can always be solved, because, after all, it will always be possible to obtain the necessary spreading or dissemination of the coal to ensure the desired contacts between all coal particles and the hot gases. However, the best mechanical solutions will give the best safety of operation, and the largest capacity per area or volume of machine.

But differential types of dryers only are adapted to the drying of a wide size ratio of feed. The largest sizes of coals (stoker, stove, and so on) require careful handling to avoid degradation, but as we all know, they do carry a smaller proportion of moisture and require less drying time than the small sizes.

Drying Characteristics

Before the proper layouts and selection of equipment can be made, we must first of all determine the drying characteristics of the coal to be treated; in other words, establish the relative drying difficulties of the coals or sizes of coals to be treated.

These preliminary studies can be done easily even in the most elementary laboratory. Our method consists in plotting the drying curves of the tested coal, just as washing curves are established for a coal cleaning problem.

Drainage factors are established first; the coal fully saturated with water is allowed to drain over a certain period of time. If, for instance, a certain size of coal is drained down to 6 pct in one hour, its drainage factor will be 1; if a slack coal is drained down to 10 pct in 12 hr, the drainage factor will be 12. In each case the final moisture contents considered are based on practical experience, coal characteristics, and actual requirements.

Drying factors are established in the same manner; coals are allowed to dry in a heated laboratory under certain constant conditions of evaporation. If it takes 3 hr to dry a certain size from its final point of drainage, say 5 pct moisture, to the required point of drying, say 2 pct, the drying factor of this coal will be 3.

If it takes 20 hr to dry a certain slack from say 10 pct of moisture, down to 2.5 pct of moisture, the drying factor of this coal will be 20. It must be well understood that this drying factor is controlled by a combination of the coal and size characteristics as well as by the particular requirements of our individual drying problem.

As an illustration the same slack as illustrated above, but with a required drying down from the same 10 pct, to say 5 pct (instead of 2.5 pct), may have a drying difficulty of 10 instead of 20; which illustrates the well known increased difficulty of drying the last 1 pct of moisture.

On the basis of such information, and of the comparative knowledge of his dryer, the drying engineer will then be able to determine all the factors required for a proper design of the complete drying plant; from the "drying characteristics" of the sizes of coal to be dried within definite moisture contents, he will deduct the "time element" of each individual problem; and by proper reference, he will be able to determine the number, sizes, and capacities of the driers required.

General Engineering of a Coal Drying Plant

The preliminary planning of a coal drying plant is a most essential part in the general planning of the whole coal preparation plant; in some cases it may be the dominating factor in the selection of the cleaning methods and apparatus.

We may, for instance, have to deal with a very slacky raw coal, with a free moisture content between 4 and 5 pct. This coal is easy to clean and could be handled satisfactorily over air tables if dried down 2 or 2.5 pct; cleaned by a wet process, this coal will dewater down to 18 pct; mechanical dryers or drainage bins will reduce the moisture down to 12 pct, and a very elaborate heat drying will be required for its final drying down to 2 or 3 pct.

Obviously, predrying followed by dry cleaning will be the cheapest solution and unless heat drying is to follow the wet process, dry cleaning may result in a higher value coal. For instance, a final product at say 8 pct ash and 2 pct moisture, total impurities 10 pct, is a better value than the same coal at 6 pct ash and 6 pct moisture. Furthermore, in Canada and some of the northern states, coal with a free moisture content of 6 pct will freeze solid in the winter time.

When considering a wet process, the determining factor in the choice of the apparatus to be selected and of the flowsheet to be adopted, is the true relationship between ash and specific gravity over the whole range of sizes.

Coals with wide "dispersions" are less suited to "bulk washing;" better results will be attained in their case with a flowsheet providing for presizing, followed by washing in small independent units.

As an illustration, a size density analysis of a well known coal from the LaSalle County, Ill.,* is given in Table 3.

Table 3 . . . Size Density Analysis Mine A, Coal No. 5
Gravities 1.30 to 1.35

Sizes, Inches	Weight, Per Cent	Ash, Per Cent
5 to 3	17.90	15.55
3 to 1.5	26.70	12.45
1.5 to ¾	16.15	13.36
¾ to ¼	25.40	10.75
¼ to ⅛	32.54	8.62
⅛ to 0	27.70	5.80

These gravities are obviously an important part of this coal, an average of 25 pct; still the same gravities indicate variations in ash content of from 5.80 to 15.55 pct. Many coals in Alabama, Washington, and Canada indicate a still wider dispersion.

This short discussion indicates that the drying and cleaning characteristics of the coal to be treated must be taken into equal consideration when making the preliminary investigation for a coal preparation plant.

Practical Conclusions

As a conclusion, our drying problem may be essentially:

1. Predrying of the raw coal in conjunction with a dry cleaning process: Or, following a wet process:

2. drying of bulk coal,
or

3. drying of presized coal,
and a fourth solution, common in Europe will be

4. predrying of the raw coal, to allow efficient dedusting, followed by:

   a. wet processing of the dedusted coal

   b. drying of the processed coal.

The present systems for the cleaning of fine coals are still inefficient, the drying of the fines is difficult and expensive; dedusting of raw coal will result in savings and improved final results.

The next step will be the selection of dryers, their number, and their general arrangement.

We hope the technical discussions of our preceding paragraphs may help for a better understanding of the problems involved, and of the solutions required.

Generally speaking, most dryers can be mechanically adapted to the drying of fine coals; but their efficiency will depend on their ability to ensure the best possible contact between coal and hot gases (only type of dryer considered), and on that basis, the dryer capable of operation at the lowest temperature will be the safest and most efficient to operate. But its initial cost may be higher because of the necessity of using larger fans to handle the correspondingly larger volumes of gases required.

Predrying of bulk raw coal to condition it for a dry cleaning plant, or for preliminary dedusting, will require special types of dryers to minimize degradation.

Drying of wet treated bulk coal will require a selective type of dryer; as an illustration, in the same feed the large sizes have a difficulty equal to 5, the small sizes a difficulty equal to 20, and unless the dryer is adapted to retain the small sizes four times longer than the large sizes, the drying will not be uniform and may prove entirely inadequate. In this case also the mechanical features of the dryer must avoid degradation.

Drying of sized coal will be the easiest problem, but unless we deal with a large capacity plant, it may require too many small capacity units, at an excessive initial cost of installation.

In all cases where substantial outputs are required, dryers with the largest unit capacity will be cheaper to install and to operate. The cost of the dryer itself may be from 10 to 20 pct of the total cost of the drying plant; the more dryers, the more conveyors, elevators, fans, and so on.

Finally, in some cases, the proposed drying plant may be located in the vicinity of a steam plant, coke ovens, or some heat process plant, disposing of wasted gases at temperatures between, say, 250° and 350°; obviously in that case a dryer capable of operation at such low temperatures will be the most desirable solution.

General Conclusions

We have endeavored to restrict this paper purely to technical discussions of the problems involved in coal drying, and to an exposé of some of the practical methods we have evolved in this connection in the course of 40 years of work in Europe, Canada and the United States.

Coal drying is still relatively a new art in this country, but its importance is growing as the cleaning of the fine sizes of coal is progressing and taking a larger importance.

Acknowledgment

We wish to express here our appreciation for one of the first material contributions recently published by the AIME, namely "The Thermal Drying of Fine Coals," by Orville R. Lyons, Research Engineer, and A. C. Richardson, Supervisor, Battelle Memorial Institute, Columbus Ohio. We believe this valuable work sponsored by Bituminous Coal Research, Inc. has opened the road to a true scientific approach of the subject, and should result in further efficient developments; in the above paper we have only endeavored to bring our personal contribution.

* From University of Illinois *Bull.* 217, 28 (13) (Nov. 25, 1930).

Thickening—Art or Science?

By E. J. ROBERTS,* Member AIME

Prior to 1916, thickening was an art, and any accurate decision as to what size of machine to install to handle a given tonnage of a specific ore must have been one of those intuitive conclusions, based on both intimate and extensive acquaintance with thickners and ore pulps.

Then in 1916 "knowledge of acquaintance," became "knowledge about" with the publication of the Coe and Clevenger paper.[1] The unit operation of thickening had graduated to the status of an engineering science. The fundamental similitude relationships for the two major phases of the operation were defined so clearly that batch tests on models as small as liter cylinders could serve to specify protypes as large as 325 ft in diameter.

It is quite apparent from reading the literature that Coe and Clevenger's contribution is not generally appreciated. In so far as the basic engineering relationships are concerned, the only real advance which has occurred in the 30 odd years which have elapsed since the Coe and Clevenger paper is the recognition of the effect of the rakes on the thickening process. Bull and Darby[2] noted this in 1926, and the extensive use of the "gluten type" thickener, in which the effect is magnified, bears witness to its importance. Comings[3] further verified this effect of the rakes.

As a matter of fact, a number of papers show an apparent regression from the Coe paper in that the area determinations are made on the basis of a single test from one concentration of solids. Coe and Clevenger amply demonstrated that this is unsafe, since the controlling zone may be one other than that of the feed dilution. Comings[3] neatly demonstrated this without apparently realizing it.

Of course there have been significant advances in the application of the operation to industry. Open tray thickeners were introduced to save area; balanced tray thickeners, washing thickeners, and multifeed clarifiers were developed with all of their special hydraulic and mechanical problems. Combinations of all kinds have been introduced, such as combination agitators and thickeners, combination flocculators and clarifiers, combination thickeners and filters.

With the establishment of the operation on a firm engineering foundation, installation was facilitated and expansion proceeded.

There are still problems, of course, functional as well as mechanical. Sometimes the moisture in the underflow obtained in practice is not as low as is expected on the basis of the test data. Sometimes the underflow is so "thick" that its discharge and subsequent handling requires special attention. Island formation plagues some operators. The use of the thickener as a surge basin and blending tank in the cement industry poses unusual problems. Design of rakes and the drive mechanism must be continually improved. Corrosion problems must be overcome. Power requirements for raking the settled solids occasionally is the controlling factor as it was in the case of the all American Canal desilting installation.

New York Meeting, February 1948. TP 2541 B. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before June 1, 1949. Manuscript received at the office of the Institute Dec. 9, 1948.

* Director of Research, The Dorr Co., Inc., Westport, Conn.

[1] References are at the end of the paper.

Other similitude relationships and design problems come into the picture when we enter the field of clarification or nonline settlement. We have an energy dissipation problem in introducing the feed and any models must satisfy the Froude model relationships. Autoflocculation requires detention which involves the same similitude laws that we encounter in the compression zone.

Approach to an Exact Science

The next step beyond having control of the similitude relationships is to understand the why of these relationships right back up the line to first principles. The ultimate might be that, if given the mineralogical composition of the solids and their size distribution together with an analysis of the suspending liquid, we could calculate the entire thickening behavior of the system. Then we could say we had reduced the operation to an exact science. True it might be more trouble getting this basic analytical data than to make our empirical determinations for area and volume, and we would need an ENIAC to calculate the results, but that does not detract from the desirability of such understanding.

Considerable work has been done by the chemical engineers with this end in view. Comings,[3] Egolf,[4] Work,[5] Kammermeyer,[6] Steinour,[7] and others have studied the problem. The writer has no final answer to the thickening story but would like to propose a picture of the mechanics of the two phases of thickening which has been found useful in understanding the subject and which leads to some convenient relationships in treating the compression step and arriving at the compression depth.

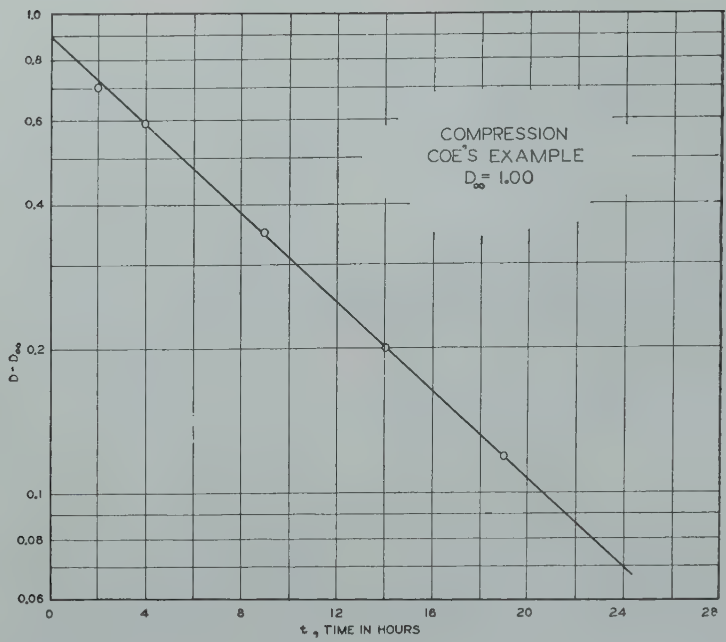

FIG 1—Compression, Coe's example.: $D\infty = 1.00$.

Teeter Conception

As a result of a study of the fundamentals of hydraulic classification it was found that fine spheres in teeter in the low Reynolds number range obeyed the following equation:

$$v = 7\frac{(ds - 1)d^2}{n}\frac{Fds}{\left(1 + \frac{1}{Fds}\right)^2} \quad [1]$$

Where: v = velocity of water at surface of teeter bed.
ds = density of spheres.
d = diameter of spheres.
F = dilution, parts water per part solids by weight.
n = viscosity of water.
(all in centimeter-gram second units.)

For water at 20°C and with v in feet per hour and d in mm for convenience, we have:

$$V = 826(ds - 1)d^2\frac{Fds}{\left(1 + \frac{1}{Fds}\right)^2} \quad [2]$$

Steinour[7] arrived at a similar equation (but not in terms of F) for sedimentation of particles.

Now, if a uniform bed of sand is in teeter at velocity v and the water is suddenly shut off, the surface of the bed subsides at this same velocity v, until the surface hits the bed of accumulated sand, and then the velocity becomes zero. This occurs at a percentage of solids by volume of about 45 pct or where $\frac{1}{Fds + 1} = 0.45$. A similar behavior occurs at higher or lower velocities (higher or lower F values). If F is carried to such a value that the percentage of solids by volume is less than about 10 pct, $\frac{1}{(Fds + 1)} = 0.10$, there is no longer a sharp line of demarkation at the top of the teeter column just as when a pulp is diluted excessively it does not settle with a sharp line. The analogy with a Type 1 pulp is very close except that the sand does not compress.

Therefore, it is proposed that a Type 1 pulp be considered as a subsiding teeter column of discrete, uniformly sized flocs. Any sand not entrapped by the flocs will segregate and settle out ahead of the flocs. Fine sand may be entrapped by individual flocs because of their "micro-yield value"[8] and larger grains may be held up by several flocs; especially at low dilutions.

If the system obeyed Eq 2 over the whole free-settling range, a test at one dilution would be sufficient but unfortunately this is not true. The effective diameter of the flocs changes with F, and therefore area determinations at various dilutions are still necessary.

To illustrate the point let us consider Coe and Clevenger's example of a Type 1 pulp. They do not give the critical dilution but from their Fig 1 it is estimated to be 3.15 to 1. If the behavior is similar to that of sand, we may estimate the composition and density of the flocs.

1000 cc of 3.15 to 1 pulp =	1179 g
Solids	284
Total water	895
Free water	550
Floc water	345

Hence flocs themselves have a dilution of 1.215:1 and a density of 629/450 = 1.40.

Let F' = weight of free water divided by weight of flocs.

Then at 7 to 1 overall dilution

$$F' = \frac{7 - 1.22}{2.22} = 2.60$$

$$\text{Hence } \frac{F'ds}{\left(1 + \frac{1}{F'ds}\right)^2} = 2.24$$

V = 0.79 fphr from plot; hence using Eq 2

$$d^2 = \frac{0.79}{826 \times 0.40 \times 2.24} = 0.00107$$

d = 0.033 mm or 33 microns.

Similarly at 4:1 dilution where v = 0.414 fphr

$$F' = \frac{4 - 1.22}{2.22} = 1.25$$

$$d^2 = \frac{0.414}{826 \times 0.40 \times 0.71} = 0.00176$$

d = 0.042 mm = 42 microns

In other cases the diameter was lower at lower dilutions than at high.

Anyone pursuing this phase of the thickening problem further is referred to Steinour's articles.[7]

Logarithmic Behavior in Compression

Turning our attention now to the compression zone a result is obtained which is quantitatively much more satisfactory than the above treatment of the free-settling zone.

The shape of the compression curve suggested a logarithmic decrement type of behavior. The rate curve would then be:

$$\frac{-dD}{dt} = (D - D\infty) \quad [3]$$

where D is the dilution at any time t.
$D\infty$ is the dilution at infinite time.
k is the rate constant.

This means that the rate of elimination of water from the compression pulp is at all times proportional to the amount left which can be eliminated up to the infinite time under the conditions of stirring, and so on, imposed.

Integration of Eq 3 gives:

$$\log_{10}\left(\frac{Do - D\infty}{D - D\infty}\right) = \frac{kt}{2.3} \quad [4]$$

$$\text{or } D - D\infty = (Do - D\infty)\, e^{-kt} \quad [5]$$

where Do is dilution at zero time.

It is apparent from Eq 4 that a semilogarithmic plot of $D - D\infty$ vs. t should give a straight line. Numerous plots of existing data verify the assumption, because by choosing reasonable value for $D\infty$, straight lines are obtained, well within the accuracy of the data.

Fig 1 shows a plot of the data used by Coe and Clevenger in their example of the compression calculation. The plotting has to be done by trial and error since $D\infty$ is unknown. A guess is made as to what value to use for $D\infty$ and the plot drawn. If it curves down at the right end, a lower value of $D\infty$ is chosen and the plot redrawn. In this instance the second choice was $D\infty = 1.00$ and all of the points except the first fall on a straight line very nicely and this first point is out only 0.025. From the plot, k is calculated by taking 2.3 ÷ the time it takes for $D - D\infty$ to drop to a tenth of its value. In this case it took 21.7 hr to drop from 0.9 to 0.09 for $D - D\infty$. Hence $k = 0.106$.

The required compression depth may be readily calculated by means of equations developed from Eq 5.

The volume V of compression pulp for which detention must be provided is:

$$V = V_1 + V_2 + V_3 \text{ cu ft per ton of solids per day.} \quad [6]$$

Where

$$V_1 = \text{volume of solids} = \frac{T}{24} \times \frac{2000}{62.5\, ds} = 1.333\, \frac{T}{ds} \quad [7]$$

where T is the number of hours required to reach the underflow dilution desired, Du.

For these calculations, Do must be the critical dilution, Dc; that is, $Do = Dc$. The determination of Dc will be discussed later.

V_2 = volume of water associated with solids at dilution of infinite time = $1.33\ T\ D\infty$ [8]

V_3 = cumulative volume of water over that of $D\infty = 1.33 \int_0^T (D - D\infty)\, dt$ [9]

Substituting Eq 5 in Eq 9,

$$V_3 = 1.33\ (Do - D\infty) \int_0^T e^{-kt} \quad [10]$$

$$dt = 1.33 \left(\frac{Do - D\infty}{-k}\right) [e^{-kt}]_0^T$$

$$= 1.33\, \frac{(Do - D\infty)}{-k} \left[\frac{D - D\infty}{Do - D\infty}\right]_{Do}^{Du}$$

$$= \frac{1.33}{k}\,(Do - Du) \quad [11]$$

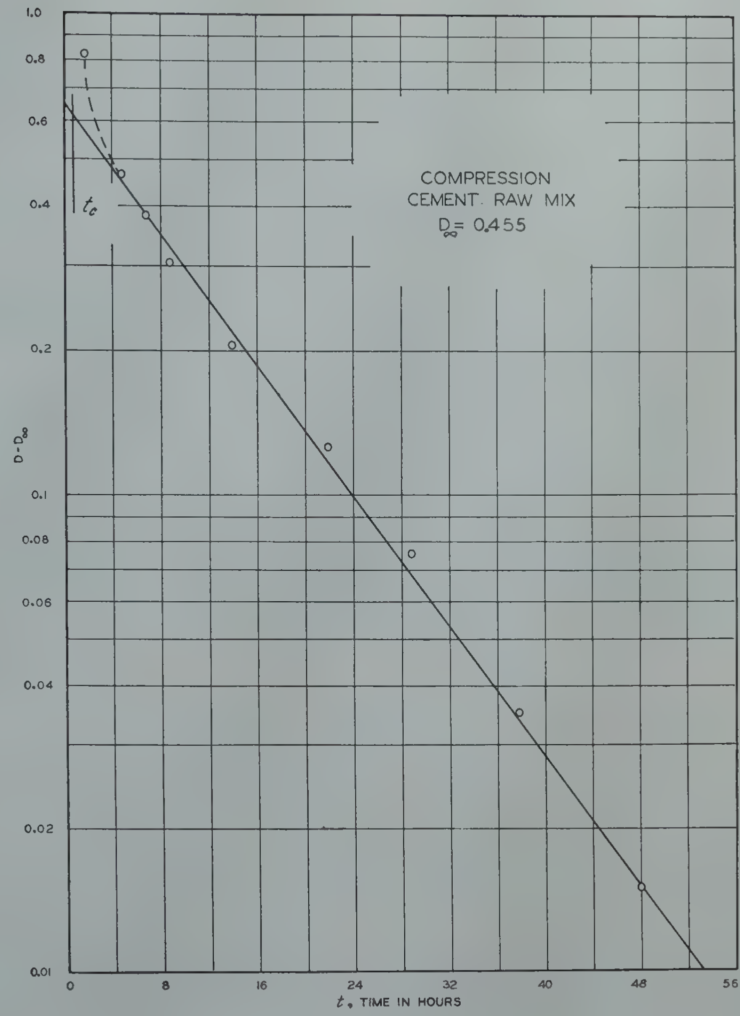

FIG 2—Compression, cement raw mix. $D\infty = 0.455$.

Thus

$$V = 1.33\ T \left(D\infty + \frac{1}{ds} + \frac{(Do - Du)}{kT}\right) \text{ cu ft} \quad [12]$$

In the Coe, example

$$V = 1.33 \times 19 \left(1 + 0.37 + \frac{0.78}{19 \times 0.106}\right) = 44.5 \text{ cu ft}$$

Since the unit area is 4.31 ft², compression depth = 10.32 ft as against Coe's 10.25 ft.

Theoretically, it should be possible to arrive at k and $D\infty$ directly rather than by trial. If Eq 3 is differentiated with respect to D, one gets:

$$\frac{d}{dD}\left(\frac{-dD}{dt}\right) = k \quad [13]$$

Therefore if D is plotted directly against t and the slope $\frac{(-dD)}{dt}$ determined at the two ends of the curve, k should result if the difference between the $\frac{-dD}{dt}$ values are divided by the difference between the two corresponding values of D. The difficulty is that the data used is not ordinarily accurate enough for this method to work satisfactorily. With a little practice, the semilog plot method is actually easier and far more satisfactory.

We are assuming, along with Coe, that the test was started with pulp of exactly the critical dilution; that is, 1.9 to 1. If the test is started with pulp in the free settling zone, a break in the curve will occur in the upper reaches of the curve, as in Fig 2. The dilution at the break-point will not be the true "critical" dilution; that is, the dilution at which the settling solids enter compression. It will be lower by the amount the first solids to settle have compacted before all of the solids have entered compression.

If the true critical dilution can be

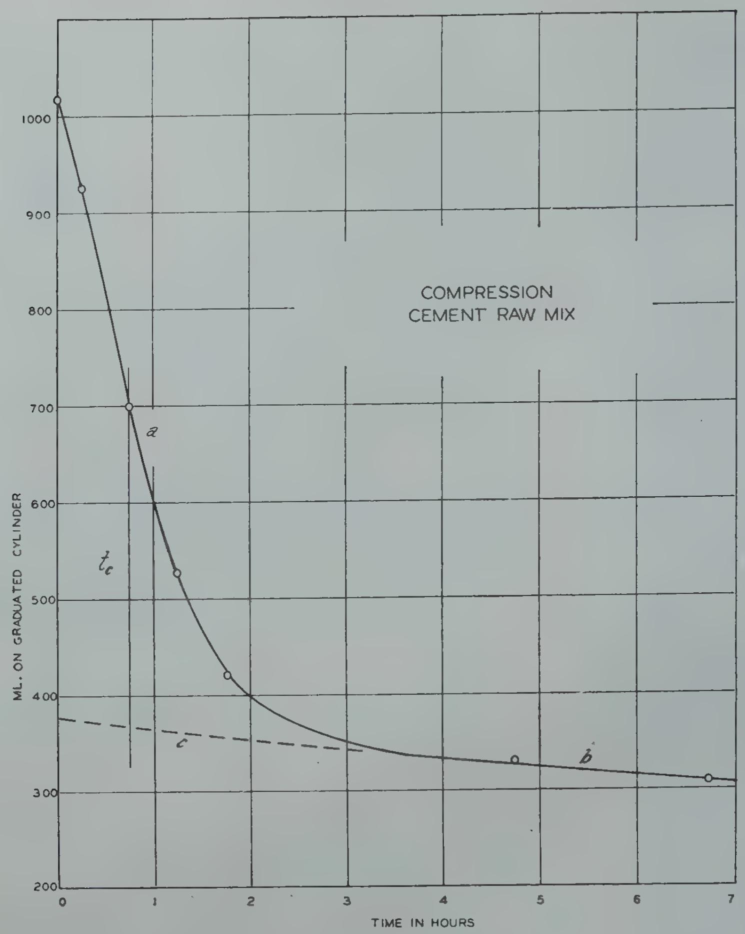

FIG 3—Compression, cement raw mix.

estimated, then the time scale can be displaced so that T, as it is used in Eq 7 to 12 inclusive, is actually the time required for the pulp to pass from the true critical dilution to the underflow dilution. It is probably not essential that this be strictly accurate and a good approximation can be obtained from the rectangular coordinate plot of the compression test (as in Fig 3) in conjunction with the semilog plot (Fig 2). The compression curve b (Fig 3) is extended (curve c) to the zero time axis with the help of the extrapolation of the straight line part of the semilog plot. By inspection, a time tc is found where the ordinate of curve a is just halfway between that of curve c and the pulp height at zero time. The result in this case is ¾ hr. This means that, in effect, all of the pulp went into compression at this time. Actually, part went into compression sooner and already has thickened somewhat, and part did not go into compression until later. The dilution of compression, Dc, corresponding to tc can be read quickly from the semilog plot at tc. In this case $D - D\infty = 0.62; D\infty = 0.455$; hence $Dc = 1.08$.

The above method is merely suggested for approximating tc and Dc. It may not work in all cases, but it saves a long and laborious calculation and is probably accurate enough for all practical purposes.

Summary

Thickening has long held the status of an engineering science but it is a still long way from being an exact science. On the other hand there is considerable "art" involved in deciding certain points; namely, what safety factor to specify to take care of possible changes in feed characteristics; whether to use the compression depth indicated or to increase the area to get a lower compression depth; how to prophesy island formation; what to do about island formation if it does occur; what speeds to use on thickeners, and so on. Careful, planned observation and reporting on the part of operators is needed before some of these items can be reduced to the engineering science status.

The hypothesis is advanced that the rate at which water is eliminated from a pulp in compression is at all times proportional to the amount left which can be eliminated up to infinite time. The use of this concept in calculating compression depths is demonstrated.

It is proposed that Type 1 pulps in the free settling zone be considered as a subsiding teeter column of discrete, uniformly sized flocs.

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Alluvial Tin Mining in Malaya

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Introduction

A relatively small area in Malaya, about 200 miles long by 40 miles wide, is the most important source of tin in the world. Some tin is recovered in other parts of the peninsula. Of the tin mined, 98 pct is recovered from alluvial deposits. From 1935 to 1941 the average annual world production of tin was 190,000 tons. The average annual production from Malaya during the same period was 62,000 tons or about one third of the total. Other producing countries, in order of importance, were the Netherlands East Indies 34,000 tons, Bolivia 30,000, Belgian Congo, Nigeria, Siam, Burma, China, and a few others with smaller amounts. The serious shortage of tin during the war period was due to the fact that the Japanese were occupying Malaya, Netherlands East Indies, Siam, Burma, and China which, together, formerly were producing 65 pct of the world supply.

Because of the importance of tin in the world economy, and because of the fact that little is generally known in this country regarding the methods of recovering the tin in Malaya, it is believed that a short description of the industry, as carried on there, may be of interest to many of the readers. In addition you may be interested in the effect of the Japanese occupation, subsequent recovery of the properties and conditions existing at that time.

The Malay Peninsula lies at the southeast tip of Asia. Singapore, a British colony, is situated on a small island at the southern end of the peninsula. It is a city of nearly a million population, an important world port, the site of a large British naval base and of one of the tin smelters. The Federation of Malaya consists of nine states, plus Penang, Malacca, and other small areas formerly part of the Straits Settlements, and it now covers the entire peninsula. The civil administration is controlled by the British. A constitution was adopted recently after reaching agreement with the Malay Sultans of the various states, who retain control of religious matters. The Capital is at Kuala Lumpur, a city of about 200,000 population, 250 miles north of Singapore and about half way to Penang. The country covers an area of 51,000 square miles, an area about equal to that of Florida and about one third that of California. It has a population of about seven million consisting of Malays, Chinese, East Indians, and a relatively small number of Europeans. The principal products are tin and rubber. There are over three million acres of planted rubber. The country is not self-supporting agriculturally as about one third of the necessary food supplies is imported. Rice is the principal food of the Asiatic population and some of the recent unrest is caused by a shortage of rice and the consequent high price. Good highways have been built through the settled part of the country. A railway, Government owned, extends from Singapore northward through the entire peninsula. The climate is tropical but health conditions are good for a tropical country.

I was in Malaya from 1939 until January 1942, during the time of the simultaneous air attacks on Pearl Harbor, Manila, Hongkong, and Singa-

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FIG 1—Early Chinese mining, stripping and mining by hand methods.

FIG 2—Early Chinese mining, tin bearing material mixed with water being bailed in stages from the bottom of the pit to the sluices on the surface. This method has been replaced by gravel pumps.

pore, the landing of the Japanese at the north, the sinking of the British warships off the coast of Malaya, and the subsequent abandonment of the mining properties as the Japanese offensive progressed southward. I returned to Malaya in October 1945, initially as a member of the Tin Inspection Committee for the British Government, assisting in making a survey of the entire tin industry, and remained until the end of 1947 in active charge of the rehabilitation of the equipment of the company with which I was employed.

History

Some tin was mined in Malaya prior to 1500. When the Portugese captured Malacca at about that time, the Malays were using a tin coinage. When the Dutch captured Malacca from the Portugese about 1640, trading stations to control the tin trade were found near the mouths of the rivers. For 200 years very little development was accomplished. There was some fighting between the Siamese and Malays for control. Later, Chinese immigrated to the country and became active in tin mining. Fighting among the Chinese factions led to the intervention of the British in 1873. Order was established and many more Chinese immigrated in subsequent years.

The first British tin mining company was formed in 1892 to carry on open cast mining. It was not until 1912 that the first dredge was installed. Development has been rapid since that time. By December 1941, when war broke out in the Far East, 127 dredges had been constructed in Malaya of which 104 were then in operation. In addition there were over 700 mines operated by machinery and 150 smaller operations.

As in this country, when dredging was introduced, the equipment was primitive when compared to present day dredges. Recovery of the ore was accomplished by means of sluices on board. As the specific gravity of the tin ore (cassiterite) is only about 7, and some of it is fine, the saving was not efficient and the resulting concentrate was low grade. In 1922 the Yukon Gold Company, predecessor of the present Pacific Tin Consolidated Corporation, introduced the use of jigs on board one of their dredges and later on others. At first these were Hartz type jigs. These have been replaced by more modern types, primarily on account of space and weight considerations. All of the modern dredges are now equipped with jigs. Other improvements have been made which will be described later.

When the Chinese immigrated in large numbers during the latter part of the last century, they undertook tin mining on a progressively larger scale. At that time most of the work was done by hand (Fig 1). Substantial yardages were moved by thousands of coolies using baskets carried on poles and by wheel barrow. Water was often bailed from deep pits by hand, in stages (Fig 2). In shallow workings treadmills were sometimes used. A few of the early Chinese who came to the country as coolies, accumulated enormous fortunes. Their estates still own large real estate holdings in Singapore, Kuala Lumpur, and other cities.

While the development of dredging

by the European companies was taking place, the Chinese miners were developing a method of using gravel pumps which has eliminated most of the hand work at their mines, and which, now, has become almost universal with them. This will be described later.

The rapid development of tin production caused an over supply of tin during the 1920's, and on March 1, 1931, an International Tin Committee was formed and an agreement was reached between the principal producing countries to limit production so as to maintain a stable price. The Tin Committee established quotas for each quarter and the various countries limited production accordingly. To provide for minor fluctuations during the quarters, a buffer pool was established to which producers could subscribe. The management of this pool sold tin when the price went up and bought when the price was down. Quarterly quotas were sometimes as low as 25 pct of normal production and the yearly averages varied between 33 and 100 pct. The scheme was effective and was in force until the end of 1946. Up to this time no new agreement has been made, as the present world shortage does not make it necessary to restrict production.

Government

As stated in the preliminary remarks, Malaya, while not a British colony, is under British control. All mineral rights remain with the Government and mining leases are granted to approved applicants. The usual term is 21 years. Leases are transferable. Usually, when near towns, there is a condition on recent leases providing that the ground must be reasonably leveled upon the completion of mining. Work must be carried on at a specified rate each year or the lease is subject to cancellation. Leases may be aggregated, so that work done on one lease applies to others, if the others are logical reserves.

Prospecting licenses are issued covering State Land when requested. The term is usually for one year. Prospecting accomplished must be reported monthly. If payable areas are proved, leases are issued over selected areas. A premium of $50 per acre is charged as well as a yearly fee of 50 cents per acre.

The Government agency administering the mining laws is the Mines Department, headed by the Chief Inspector of Mines. He appoints a Senior Inspector of Mines for each State and each Senior Inspector is allowed a staff sufficient to carry on the duties of his office. Safety rules are enforced and periodic inspections of all workings are made to see that the mining laws are being observed.

As in all placer mining in settled areas, there is a muddy water problem. This is handled logically. Muddy water from the mining operations may be discharged to the rivers, provided it does not carry over 800 grains of solid matter per gallon. Its appearance does not make any difference.

Revenue to the Government from the tin mining industry is one of the principal sources of Government income. It is collected as royalty assessed when the ore is exported for smelting. The royalty is based on the weight of ore and a sliding scale based on the price of tin. When the price of tin is 50 cents per pound the royalty is 13.7 pct. When tin is $1 per pound it is 15 pct. This seems high but it must be considered that the ore is a wasting asset. There is no tax on equipment and up until the end of 1947 the royalty was the only direct taxation of the mining industry. At the beginning of 1948 a relatively low income tax was initiated.

Generally speaking it is considered that the mining laws are fair and that they are fairly administered.

Government is faced with a problem regarding claims for war damage. All of the operators have filed claims, under different headings such as loss of equipment, tin ore extracted by the Japanese and the cost of rehabilitation of recovered equipment. The total of these claims is far beyond the resources of the present Government and there seems little chance that sufficient reparations can be paid by Japan to meet these claims. In order to get the industry started, the British Government has made certain loans, interest free for the first few years, to be applied against war claims if and when allowed, but repayable over a period of years if the claims are not allowed.

Ore Occurrence

The main physiographical feature of the country is the granite range of mountains which extends some 300 miles long and 40 miles wide in a north-south direction near the western side of the peninsula. The crest of this range is from 4000 to 7000 ft in height. The tin deposits of Malaya are associated with the granite. To the west the granite is in contact with limestone, and it is along this contact that the richest deposits were formed. Further west, shales are the predominant bedrock.

It is the erosion of the granite and the contact zones which accounts for the extensive alluvial deposits. In most of the area the granite-limestone contacts are "frozen," but a few very rich contact deposits have been found and worked extensively by open pit methods. In some areas small stringers carrying tin ore have been found in the granite but no extensive lodes have been found in it to date. There is one large lode mine near the east coast, over 200 miles from the main placer deposits where the tin ore occurs in quartz veins in schist. To date this is the only large scale lode mine in the country.

While the occurrence of tin ore is very general, very few other minerals occur in commercial quantities. There is one lode gold property which has been worked since 1898 with a production to date of about 750,000 fine ounces of gold. In some few areas a small quantity of placer gold is recovered with the tin ore, but this is not general. Other minerals which occur in small quantities are scheelite, wolframite, manganese, and ilmenite.

Two relatively low grade iron ore deposits near the east coast have produced for several years. They were operated by Japanese before the war. In 1940 about 2,000,000 tons of iron ore were produced and exported to Japan.

Coal is found extensively in one area. The coal is geologically recent, not far removed from lignite. It is produced extensively, however, and is the principal fuel used in the country. It is used on the railways, steam power plants, and on many steam powered dredges. The coal cannot be bunkered for any length of time as it will heat and burn if stored. Some of the coal is recovered by strip mining and some from underground workings. The deeper coal seams, mined by underground methods, produce the best coal. Before the war, production was averaging 500,000 tons per year. There are reserves for many years.

To date no oil has been found and all petroleum products are imported.

FIG 3—Banka type boring outfit and crew.

Prospecting

Practically all prospecting, both by European companies and the Chinese miners, is done with the Banka type drills (Fig 3). This is because such equipment is transported easily and this is important as much of the potential mining areas are swampy. It would be impracticable to transport heavy drilling equipment such as a Keystone drill over most of the areas.

The Banka type drills consist of heavy 4 in. pipe, with sleeve couplings, the pipe being made up in 5-ft lengths. The cutting shoes are about 5 in. in diameter. Clamps with long extensions are placed near the top of the pipe. A wooden platform is placed on the clamps and four men mount the platform. 1⅛ in. square rods in 10-ft lengths, with male and female thread couplings, are used. As most of the deposits are sand and clay, most of the drilling is done with the pumps. These are 4-ft lengths of 3-in. pipe with a replaceable shoe and clapper valve at one end and a bail with male threads to fit the rods at the other end. Other drilling tools are an auger for use in stiff clay and a chisel bit for use when rocks or wood are encountered. By means of wrenches, which fit the rods loosely, the four men of the platform raise and lower the pump until it has recovered a sample. Four men on the ground turn the pipe by means of poles. The weight of the men on the platform lowers the pipe as it is rotated. At intervals, the men on the ground exchange places with those on the platform.

When the pump is filled, it is raised to the collar of the hole, the rods being disconnected in turn and the pump is dumped into a sample box on the ground. In ground averaging 60 to 70 ft deep, normal progress for a drill crew is 50 ft per shift. In deeper ground the progress is slower as the weight of the rods increases and more time is consumed in connecting and disconnecting the rods and pumps. The crews, usually Chinese, are hired through one man in groups of eight.

Each 5 ft of core is kept as a separate sample. An Asiatic clerk with each drill crew records all possible data as the work progresses. Upon completion of 5 ft of drilling, the sample is drained, sacked, and carried to a central washing plant. There, the samples are measured and then washed in a dulong (similar to a batea). This washing is done usually by Chinese women who first puddle the sample until all clay is disintegrated and then wash it until a rough concentrate is obtained (Fig 4). This concentrate is then cleaned by an expert Chinese tin washer and the clean tin ore is weighed. The value per cubic yard is then calculated. A careful log is kept of each hole for future reference. Sampling each 5 ft separately determines the vertical distribution of the tin ore.

As the samples are small, a reliable average value is obtained only when sufficient drilling has been done. Preliminary boring is often done at the corners of 400 ft squares, when large areas are being tested. If this indicates payable values, additional holes are put down. Final boring is often carried to 50 ft squares.

Recoverable values are calculated, making allowances for odd holes with abnormally high values, condition of bedrock, amount of clay, and any other features which would affect recovery. After a reasonable amount of dredging has been accomplished in an area, it is possible to arrive at a factor which makes it possible to estimate future recoveries quite closely.

FIG 4—Chinese women making rough concentrate from samples from boreholes.

Dredging Conditions

Dredging conditions vary considerably throughout the area. In general, however, the material being dredged is sand with some fine gravel with varying amounts of clay. There is very little coarse gravel as known here in California. Near the granite contact the deposits are generally underlaid by limestone bedrock and contain more clay than those further away underlaid by shale bedrock. They are largely the result of decomposition of the

granite and consist of quartz sand and clay, the clay being the result of decomposition of the feldspar. Some deposits near the contact are "in situ" and are simply decomposed granite with a stock work of small veins.

The limestone bedrock presents real dredging problems as it is often very irregular, consisting of pinnacles and crevices. In a dredging area which boring shows to have an average depth of 60 to 70 ft there may be pinnacles extending nearly to the surface. Often these are not found in prospecting but are encountered in the course of dredging. In many cases it has been necessary to drill and blast the tops of these pinnacles in order to provide safe flotation for the dredges. Depths of digging in one face may vary from just a few feet to maximum depth. Obviously all of the tin ore cannot be recovered by dredging in this type of deposit. However, tin ore, with a specific gravity of about 7, is not as definitely concentrated near bedrock as is usual in gold deposits. In estimating values, deductions are made to compensate for losses due to rough bedrock. It is estimated roughly that two thirds of the dredging is being done in areas underlaid with soft shale bedrock, and about one third in limestone areas.

Many of the dredging areas require rather extensive clearing of timber and jungle. Some have considerable buried timber.

Treatment of the clay deposits is a serious problem and will be described under operation methods.

In comparison to dredging in California, one condition is noteworthy, and that is that the wear on dredge buckets, lips, bushings and pins is much less in Malaya. This is due, primarily, to the fact that there is no heavy, tight gravel as known here. In addition, the slime resulting from washing the clay deposits acts as a lubricant to some extent.

FIG 5—Dredge 6 of Pacific Tin Consolidated Corp. near Kuala Lumpur, Malaya. Bucket capacity is $13\frac{1}{2}$ cu ft; digging depth, 80 ft; electrically driven, equipped with clay treating devices. Designed and fabricated by Yuba Manufacturing Co., San Francisco, and erected by the local staff of Pacific Tin in 1937.

Dredging Equipment

As mentioned in the preliminary remarks, there were 104 dredges in active operation in Malaya in 1941 when the Japanese over-ran the country. Practically all were of steel construction. Generally speaking, the digging end of the tin dredges is similar to that of the gold dredges in this country. The difference is in the treatment of the material on board to recover the values.

Being in British territory and subject to empire preference, most of the dredges were manufactured in England, some from Holland, and some from the United States. The majority of the more modern dredges are electrically driven but many of the older ones are driven by steam, burning Malayan coal. Some have been converted to burn oil.

The dredges vary greatly in design, size, and efficiency. Bucket sizes vary from 6 to 18 cu ft, and digging depths from about 35 to 125 ft below water. They all operate on head lines instead of spuds as is usual in the United States. This is due to the fact that the fine material does not afford sufficient backing for spuds.

The most recent dredges in Malaya are four, which were put into operation in 1939. They were manufactured in England, have 18 cu ft buckets, dig 125 ft below water and have a capacity of 500,000 cu yd per month.

Very few dredges in Malaya are equipped with tailing stackers. The fine nature of the material causes very little swell when the tailings are deposited behind the dredge. The upper tumbler centers are relatively high, providing sufficient grade enabling the use of long overhanging tail sluices. All modern dredges are equipped with jigs.

On some of the most recent dredges, there are several features of design which, as far as known, have not been used in this country. One of these is the use of fluid drive in the transmission of power from the main drive motor to the bucket line. This is intended to reduce the shocks on the main drive gearing when the bucket line encounters obstructions. Another is a device to reduce the length of the catenary in the return of the bucket line. The upper section of the ladder is increased in depth. The bucket line, upon leaving the upper tumbler, enters this section of the ladder and passes through it on a tread similar to a Caterpillar tread. Some of the newer dredges are equipped with two revolving screens, one on each side of the center line. The hopper is arranged to divide the discharge from the bucket line to the two screens, and also to divert the entire discharge to a chute installed on the center line. This chute is used when it is desired to strip the upper worthless strata without washing. Another device used is an assembly of gears in the main drive gearing, making it possible to obtain varying speeds for the bucket line by shifting gears much as is done in an automobile transmission. This makes it possible to use varying speeds for the bucket line while using a constant speed motor, and is less expensive than the Ward-Leonard control with variable speed motors. Some dredges are equipped with ladder hoists driven by worm gearing. This has the advantage over some other types in that a careless operator or the failure of complicated electrical brakes cannot cause the ladder to be dropped. Power must be applied either to raise or lower the ladder.

Dredges as described above, may be considered the standard type. They

dig the material which is then discharged through the hopper to a revolving screen. The undersize from the screen is distributed to the jigs, and the oversize is discharged aft of the dredge as tailing. In deposits containing large percentages of clay, special equipment has been developed. Some of the more modern dredges in Malaya incorporating this special clay treating equipment were designed and built in the United States. An example is the newest dredge of the Pacific Tin Consolidated Corp., completed in 1938. This proved to be outstanding in performance in extreme clay conditions requiring highly specialized equipment. Mr. George Coffey, President of the dredging company, cooperated with Yuba Manufacturing Co. in designing the dredge, furnishing from his own knowledge and experience, designs for the special equipment required to treat the large quantities of clay being dredged (Fig 5). This dredge has 13½ cu ft buckets, digs 80 ft below water, is equipped with motors totaling over 2000 hp, and has a normal capacity of 350,000 cu yd per month.

In deposits of this type, the first difficulty to be overcome was the emptying of the buckets. This was accomplished by changing the design of the buckets to a bowl shape, and introducing a clay extractor just aft of the upper tumbler. This consists of three wide blades about 3 ft long attached to a shaft at equal intervals. No power is applied to the shaft, which is held in position by three sided cams. As a bucket passes over the upper tumbler, a blade enters the bucket just inside the lip. The continued progress of the bucket forces the blade into the bucket where it scoops out the contents, and the next blade is brought into position to enter the next bucket. The clay extractor is suspended so that it can be moved back out of service when it is not required.

On what has been called the normal dredge, material of this nature would not be broken up and the tin ore which it contains would not be recovered. On these special clay treating dredges the material is passed, first, through the ordinary revolving screen to remove any fine material not requiring further disintegration. The reject from this first screen is then passed to a second revolving screen, equipped with manganese grid plates with 4 by 6 in. holes. A shaft is installed through the length of this screen and about 30 clay knives of ¾ by 4 in. steel are assembled on the shaft by suitable attachments. These knives reach nearly to the shell of the screen. The shaft is rotated at about 150 rpm in a direction opposite to that of the screen and the clay knives chop the clayey material until the lumps are small enough to pass through the grid plates. This material falls into two double log washers installed under the screen. The log washers disintegrate the material which is then passed to auxiliary screens and the fines are elevated and distributed to the rougher jigs. Owing to the additional wear and tear and power used, it is obvious that this method of operation involves a higher cost than normal dredging. It can be used only in deposits where the values are sufficiently high to compensate for the extra cost. Some of the deposits containing high percentages of clay do have the higher tin ore content required.

Operating Methods

The dredges are usually built in artificial pits. The hull ordinarily is built on posts but some are launched. When the hull is complete and afloat, the superstructure and machinery are installed. The dredge is then put into operation and the cut is deepened as rapidly as possible and widened to the desired width. General practice is to carry cuts as wide as possible and widths of 1500 to 2000 ft are common.

Owing to the fine nature of the material and the presence of clay, fresh water must be added to the pond continuously. One and often two slime pumps are operated from small barges anchored at the sides of the pond. The suctions of these pumps are kept 15 to 20 ft below the surface of the pond, and they pump the thick muddy water from that level to settling areas. Water for the pumps on board the dredge is taken from the surface of the pond.

As the material is dug by the buckets, it is discharged to a hopper, from which it passes to a revolving screen 7 to 9 ft in diameter, 40 to 60 ft long, equipped with screen plates with ⅜-in. holes. Water to wash the material is introduced into the screen. On what may be called the normal dredge, without special clay treating equipment, the oversize from the screen is discharged aft through a chute. The undersize is distributed to rougher jigs installed on each side of the dredge. Four-cell jigs with beds about 42 in. square are used almost universally. The jig bedding is hematite, screened to ½ in. size, iron punchings or coarse tin ore. The jig screens have slotted holes ¹⁄₁₆ to ¹⁄₁₂ in. in width. Rougher jigs are operated at 80 to 100 strokes per minute with a 1 to 1¼ in. stroke. The concentrate from the rougher jigs is carried by launder to cleaner jigs. These are 4-cell jigs, usually operated at a somewhat faster rate and with shorter stroke. If operating properly, the concentrate from the first cell of the cleaner jigs should be 40 to 60 pct tin ore. This is taken ashore for further cleaning. The concentrate from the lower three cells of the cleaner jigs is returned to the circuit. About 40 to 50 cu yd per hour are passed over each of the rougher jigs and tests of the jig tailing indicate a satisfactory saving. In many cases tests of the jig tailing are made hourly as a check on the operators. A high tailing sample indicates clogged beds, insufficient water, or other cause, which is then investigated and corrected. The whole operation is largely automatic when it is considered that the tin ore extracted from, say, 10,000 cu yd per day is discharged continuously from ⅜ in. holes in the spigots from the hutches of the first cells of the cleaner jigs. The main problem is a mechanical one of keeping all of the equipment in operation continuously, as, unlike some other operations, all of the equipment must be in operation or the dredge cannot be operated.

The concentrate taken ashore is transported to a tin shed for further cleaning. If the concentrate contains pyrite or arsenopyrite, as is often the case, it is roasted before further washing. The final cleaning of the ore is done by expert Chinese tin washers on contract. The concentrate is washed first in short sluices to remove the lighter impurities and then, finally, cleaned by hand jigging in screens about 10 in. in diameter. Some mechanical aids such as hindered settling devices and magnetic separators are used by some companies, but a very large proportion of the operation is by hand. This cleaning could all be done mechanically but it is doubtful if either the cost or the grade of product could be improved. In addition there would be a greater cost for equipment and skilled mechanical help would be required to keep the additional equipment in operation. Pure cassiterite

contains about 78½ pct metallic tin. Shipments average over 75 pct tin and some have been as high as 77 pct. When finally cleaned the ore is dried, sacked, and shipped to the smelter either at Singapore or Penang, where it is sampled, weighed and finally smelted. Returns are received in about 10 days. High grade shipments are desirable as freight is charged on the weight, duty is based on an assumed 75.5 pct tin content and smelting charges are more favorable on high grade ore.

The dredge crew for most of the modern dredges, consists of a European dredgemaster, and three shifts, each consisting of a European shift engineer and 10 to 20 Asiatics depending on the size and complexity of the dredge. Outside crews, employed on the day shift only, consist of shore gangs to handle lines, shift slime-pump pipe lines, erect electric transmission lines, and transport tin ore ashore. In addition there are electrical and mechanical superintendents, office and warehouse staffs. Most of the larger companies have their own machine shops and several have their own electric power plants. Operations are carried on 24 hr per day and seven days a week.

FIG 6—Typical Chinese gravel pumping operation.

Economic Factors

As with base metal mining in the United States, one of the primary factors affecting the tin industry is the price of the product. Others, of course, are the cost of labor, power, and supplies. Since 1915 the price of tin at New York has varied from a low of about 20 cents per pound in 1931 to the present all time high of $1.03 per pound. During the war period the price was controlled at 50 to 52 cents per pound.

Prior to the war the average content of the ground being dredged in Malaya was ⁴⁄₁₀ of a pound of 75 pct concentrate per cubic yard, or ³⁄₁₀ of a pound of tin. Costs at that time varied from 4 to 8 cents per cubic yard, depending on the capacity of the dredge, efficiency of the operators, and dredging conditions on the various properties. Since the war, costs have risen about 100 pct. As long as the costs do not rise appreciably higher and the present or a higher price of tin prevails, the situation is favorable for the miner. For instance, if we assume a prewar cost of 5 cents per cubic yard and the value of the product of 10 cents, the margin of profit was 5 cents. On the same operation today, the cost would be 10 cents, the value of the product would be 20 cents and the margin of profit would be 10 cents, or double the prewar profit.

During the period before the war, up to 1939, the investment in equipment for what may be considered the average installation was about $1,000,000. This would include a modern electrically driven dredge with, say, 13 cu ft buckets, digging 70 to 80 ft below water, erected ready to operate and the auxiliary equipment required consisting of tools, camp for staff, tin shed, store rooms, power lines, and so on. It would be installed, probably, on an area of 200 to 400 acres containing 30,000,000 to 40,000,000 cu yd of dredgeable material. This ground might cost anywhere from $150 to $1,000 per acre, depending on how the leases were acquired. Today, the cost of such an installation would be about double the prewar cost. Even this would not be prohibitive, considering the possible profit, if one could be assured that the price of tin would remain at or above the present figure for the next 15 or 20 years. With this uncertainty in view, one can readily understand why the operating companies have been concentrating their efforts on rehabilitating the equipment they have on hand.

The ore produced in Malaya must be smelted at the local smelters in Singapore and Penang. An additional duty of about $250 per ton is levied on ore shipped elsewhere, which makes it prohibitive. The smelter accepts shipments of ore, samples and assays to determine the grade, pays the duty, freight, deducts smelting charges and penalties, if any, and remits the net return. The smelter then disposes of the tin, at present, through the Ministry of Supply in England.

Chinese Gravel Pump Mining

As mentioned before, the Chinese did the first extensive tin mining in Malaya. At that time the work was done largely by hand with very primitive equipment. During the course of the several years before the war, they developed a method of mining by gravel pumps which is now almost universal with them (Fig 6). The method may be described as follows: water, either from a near by stream or as return from the tailing area, is led to a fresh water pump near the mine. This pump acts as a booster pump to develop pressure. The water is led to three or four small monitors with about 3 in. nozzles, set in the pit. The water is played against the banks and washes the material down to a sump. An 8 in. gravel pump, often with a suction head of as much as 25 ft, and pumping against a head of as much as 150 ft, is driven by electric motor or Diesel engine. The water carrying about 15 pct solids is elevated to a sluice supported on a pole trestle high enough above the surface of the ground to provide tailing room. Usually there are two sluices about 6 ft

FIG 7—European owned hydraulic tin mining operation.

wide. The discharge from the gravel pump is directed over a grizzly to remove coarse material, and then to one of the sluices. At intervals of about 5 ft, 2 in. square strips or riffles are laid across the sluice. As the water and material flow through the sluice, a rough concentrate forms behind the riffles. As this builds up, additional riffles are added until the concentrate bed is 6 to 8 in. deep. The flow is then directed to the alternate sluice and the first sluice is cleaned up. The operation is carried on 24 hr a day with about 10 men on a shift.

It is usually necessary to impound the tailings. This is accomplished, in some cases, by depositing them in a former pit, and in other cases by impounding them on level ground. The method is to throw up a small levee around sufficient area. As the material is deposited, two or three men, using a tool like a hoe, keep raising the levee, so that it is kept about a foot above the level of the tailings. The levees are raised with a steep slope on the outside. Long grass, cut near by, is folded into the sand at 1-ft intervals and this helps bind the sand and prevents washing down during the heavy rains. Dams of this nature 50 ft high are common and they last for years.

The gravel pumps are usually 8 in., equipped with an open runner and a liner or replaceable shell. Many of them are made in the local Chinese foundries. Worn runners, liners, and shells are turned in as scrap.

The capacity of one of these operations is about 25,000 cu yd per month and the cost per yard about double that of dredging. There are inherent inefficiencies in the operation. One of these is due to the fact that material from near the surface is washed down to the sump, the lowest point in the pit, and then re-elevated to a height above the surface. The inefficiencies and higher cost are compensated for to some extent by the fact that the method recovers the tin ore more completely than dredging, where the bed-rock is rough, uneven limestone, since the crevices can be cleaned out much more thoroughly.

In 1941 there were 733 mines of this nature in operation and they were producing about one third of the tin produced in the country.

Other Alluvial Mining

In addition to dredging and gravel pump mining, which produce most of the tin, other methods are used. There are several properties operated by hydraulic methods, some on a large scale (Fig 7). There are several large open cast mines, working on contact deposits. These were worked by hand methods in the beginning but they have since been mechanized to a large degree. Stripping is done on benches, with power shovels, loading trains of cars pulled by small Diesel locomotives. Bottom material is hoisted up inclines, in cars, to washing plants on the surface. Some of these mines are about 200 ft deep now and the bottom of the workings is below sea level.

Licenses are issued to "dulong" washers, mostly Chinese women. These would correspond to "snipers" in this country. They are supposed to work on public lands only, but will encroach onto privately owned leases at every opportunity. They account for quite a sizeable production. Some of this is undoubtedly "high grade" stolen by their men folks from the dredges and gravel pump mines.

In 1940, the year of maximum production, when 80,500 tons of tin was mined, the production by the various methods of operation was as given in Table 1.

Table 1 . . . Production of Tin by Various Methods, 1941

Method	Tons
Dredging	42,200
Gravel pumping	28,300
Hydraulicing	3,025
Open cast	3,100
Underground	2,800
Dulong washers	1,000
Miscellaneous	75
Total	80,500

Results of Japanese Occupation

As the Japanese campaign progressed down the Malay Peninsula, the question arose as to what should be done regarding the dredging and gravel pumping equipment then in operation. As the Government, logically, did not wish to indicate that they were fighting a losing battle, no early instructions were issued. The heads of the various dredging companies met and decided to dig the dredges onto shallow benches, sink them there, and immobilize them by cutting the ladder lines, blasting the upper tumblers or otherwise rendering them incapable of operation. Many of them did this. Others took no action and much confusion resulted. As the Japanese army progressed so rapidly, some dredges were abandoned in operating condition. At a late date, Government issued orders to pull the dredges into deep water and sink them. This was not possible with the dredges which had already been immobilized. A few were sunk as directed. Very little was done to dispose of the spare parts on hand. At the Chinese gravel pump mines much of the equipment was blasted

and the mines were allowed to fill with water.

We know now that when the Japanese took possession, they attempted to carry on the tin industry much as it had been done before. They continued the Government Mines Department with the Asiatic staffs employed there. They formed mining companies to take over the dredges and operate them. A Japanese manager was appointed for each property. He assembled the former crews of Asiatics and instructed them to start operations. They took complete inventories of all equipment on the properties, including all spares and supplies. They put a value on everything, including the dredges, had maps made showing the property lines, the dredged areas and the areas remaining to be dredged. All of these data were in English and they were recovered in the Mines Department when the country was re-occupied. They raised several of the dredges which had been sunk in shallow water and put them into operation. They made many ingenious repairs to the dredges, used many of the spare parts which were on hand, and the records which were recovered showed that results were fair for the first few months. Owing to the lack of competent supervision, and particularly because they apparently did not have proper lubricants, the equipment deteriorated rapidly.

The Japanese dismantled about 20 dredges but they were mostly the older, more obsolete types, with one exception. One modern electrically driven dredge was dismantled and apparently removed from the country as no trace of it was ever found. It may have been taken to Japan or Korea. They did not destroy the remaining dredges when they surrendered.

Many leases were granted to Chinese miners for gravel pump work during the occupation. Some of these were still in operation when the country was re-occupied.

The nearest estimate is that they produced 55,000 tons of tin during the period of their occupation. They repaired the smelter in Penang and smelted the tin as it was produced. Some of this was recovered as block tin stored at the smelter. It was evident that they had been short of shipping during the latter stages of their occupation or this tin certainly would have been removed.

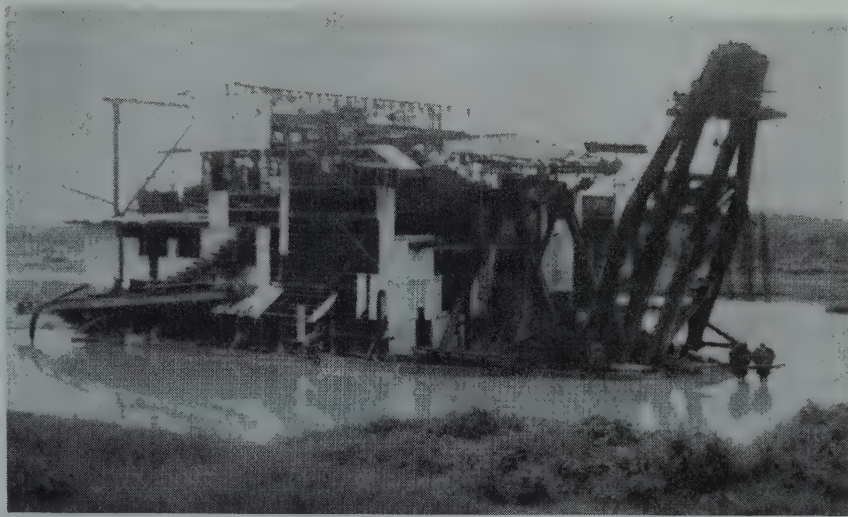

FIG 8—Dredge 3 of Pacific Tin Consolidated Corp. as recovered after the Japanese occupation. This dredge was barely afloat and all machinery badly worn. All small tools, small motors, electric wiring and house covering had been looted. The remaining housing and superstructure showed some damage from shrapnel. The dredge was completely rehabilitated and put into operation early in 1947.

Rehabilitation after the War

After the Japanese surrender in August 1945, the country was under British Military Administration until April 1, 1946, when civil government was re-established. The survey of the entire industry by the British Tin Inspection Committee was accomplished in October and part of November 1945. When this was completed, the various companies re-occupied their properties and started the work of rehabilitation. Those in charge were faced with a deplorable mess. Some of the dredges remained sunk as they had been left. Others sank during the interim period when no one was occupying the properties. Those that had been operated by the Japanese had been run to a standstill. On those, practically all bearings and gears required replacement as they had been operated without proper lubrication. Serious looting had taken place between the time of the Japanese surrender and the British occupation, a period of over two weeks. All small tools and easily portable equipment were missing. Fig 8 shows a dredge recovered after the Japanese occupation. What houses remained at the mines were completely stripped of all furniture and fittings. The gravel pump mines were full of water and much of the equipment for them had been destroyed. At first we were all living on Army rations as the regular importation of food supplies had not been started. There was little motor transport available for personal travel or for the industry. It was rather annoying to some of us to see dozens of lend-lease American jeeps lined up in an army motor pool but none available for us to use.

With the dredging companies, the first efforts were to determine just what would be needed to put the dredges into operation and to get orders placed for this material. Many items, particularly electrical equipment and steel were known to be in short supply all over the world.

The next step was to try and sort out and recover equipment which had been moved from one property to another, by the Japanese, in their attempt to keep the dredges in operation. A dredge belonging to one company was found to be equipped with a bucket line belonging to another company. Items such as machine tools, compressors, welding sets, and so on, were badly scattered. As an example I located a large gap lathe 150 miles from the shop from which it had been taken. The army furnished a tank recovery truck to bring that back.

As rapidly as possible under the conditions prevailing, the various companies started to re-assemble their staffs and start operations. The first dredge to be started was one of those belonging to the Pacific Tin Consolidated Corp. For some reason, never explained, this dredge had not been

looted, although it was tied up at the bank where anyone could board it easily. This dredge was one which had been sunk on a shallow bench and supposedly immobilized. Work on rehabilitating this dredge was started in December 1945. It started operating March 1, 1946. By the end of June 1946 there were 7 dredges in operation. On December 31, 1946 there were 18 operating, at the end of 1947 there were 56 and on June 30, 1948, there were 61. Rehabilitation work is still in progress. As far as is known, no new dredges have been built in Malaya since the war.

Rehabilitation of the Chinese gravel pump mines has progressed at about the same rate as dredging. At the end of 1947, there were 323 gravel pump mines in operation compared with over 700 prewar.

Production has increased from 8600 tons of tin in 1946 to 27,000 tons in 1947 and an estimated 40,000 tons in 1948. This compares with the average of 63,000 tons for the 7 years before the war and with 80,000 tons in 1940, the year of maximum production.

Future of the Tin Industry of Malaya

Prior to the war, a complete survey of the mineral industry of Malaya was made by Sir Lewis Fermor, a British engineer, for many years Government geologist in India. At that time he estimated that the life of the tin industry, on the scale prevailing at that time was about 20 years. It is safe to say that the known reserves of unworked alluvial deposits amount to three to four billion cubic yards. The figure is variable since there are large areas, now known, which have been considered marginal, which will become profitable if the price of tin remains at the present high figure of $1.03 per pound or goes higher. Intensive search over the years prior to the war and since, has developed additional yardage in the known districts, but no large new areas have been developed. It is unlikely that such new areas will be found.

There is always the possibility that new lode deposits may be found, as prospecting for these has not been as general as for placer deposits. It must be considered that the country is covered with heavy jungle and that, therefore, such prospecting is very difficult.

Many of the areas where dredging has been carried on over rough limestone bedrock, may be reworked by gravel pump methods when the price of tin is high.

In general, the future of tin mining in Malaya is dependent to a great degree upon the price of tin. This, in turn, will be governed to a large extent by economic conditions throughout the world, and on the production from the other substantial producers such as the Dutch Indies, Bolivia, Nigeria, and others. There is always the possibility that new discoveries may be made in other parts of the world.

Another factor which may have an important effect on the tin industry is the political unrest prevalent in Southeast Asia. This is another story, but it has caused a marked deterioration of law and order in Malaya, with the accompanying labor unrest, gang robbery and kidnapping.

Since the war in Europe started in 1939, there has been a shortage of tin throughout the world. This shortage may continue for some time, but, unless there is another war, there is not much doubt but what production will equal the demand eventually and probably pass it at some undetermined future date. Much will depend on the extent of stock piling in this country as to what that date will be. The importance of a reserve of tin becomes apparent when it is realized that, in normal times, the United States uses about 45 pct of the world's tin and there is practically none produced on the North American continent.

Drilling Blastholes at the Holden Mine with Percussion Drills and Tungsten Carbide Bits

By ELTON A. YOUNGBERG,* Member AIME

The Holden mine operated by the Chelan Division of the Howe Sound Co. is on the east slope of the Cascade Range in north central Washington on the south slope of Railroad Creek valley at an elevation of 3500 ft. The mine may be reached by a 40 mile boat trip from the town of Chelan which is at the southern tip of Lake Chelan, to Lucerne at the mouth of Railroad Creek and an 11 mile bus ride up Railroad Creek to Holden. All freight and concentrate is moved over this route to Chelan Falls on the Columbia River which is on the railroad four miles below the town of Chelan.

The mine is now producing 2000 tons of gold, copper, and zinc ore per day which is treated in the Holden mill. Gold-copper and zinc concentrates are made, the first of which is shipped to Tacoma, Wash., and the latter to Kellogg, Idaho, for smelting.

Ore is broken by long-hole blasting using the Noranda system which has been modified to meet local conditions. Until recently, blastholes have been drilled by diamond drills. Now a partial substitution of percussion drill holes, drilled with tungsten carbide insert bits, is being made.

Geology

The ore body occurs as a replacement deposit in a highly metamorphosed series of sedimentary rocks, mainly gneiss and schists, in a shear zone several hundred feet in width and of undetermined length. Commercial ore has been found in mineable widths of 25 to 100 ft for approximately 2500 ft along its strike. The commercial minerals are chalcopyrite, sphalerite, and gold. During the period of mineralization considerable silicification took place giving the ore an abrasive drilling characteristic. Following the period of mineralization, numerous dikes were introduced into the ore body. The earlier ones were of granite composition having a width of a few inches up to 80 ft. These were followed by much younger, fine grained basic dikes which usually do not exceed 2 ft in width.

Development of Percussion Blasthole Drilling Equipment

Test work with the 1½-in. tungsten carbide bit was carried on in development headings for several months early in 1947. The short life of the bits, because of gauge loss caused by the abrasive nature of the rock, prevented its adoption for this use. However fast drilling speed and ability to drill a long uniform hole suggested its use for drilling blastholes in competition with diamond drills as diamond costs were steadily increasing and experienced drillers were difficult to obtain.

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The 1½-in. bit was the largest available at the time initial test work was started with sectional steel. The 1½-in. hole limited the diameter of the steel thread and coupling which could be used. Type F couplings were first used but because of the small thread section excessive breakage of the steel was experienced. Type H couplings were tried next. In order to use this coupling which is 1⅝ in. in outside diameter, it was reduced to 1⅜ in. giving ⅛ in. clearance between the coupling and the hole. Rod breakage at the thread was substantially reduced but some coupling breakage was experienced, however the overall performance was considered satisfactory (see Fig 1 for illustration of coupling and thread). Early test work with the 1½ in. bit indicated machines of piston diameters larger than 2⅝ in. would cause inserts to loosen or break. It was found however that the additional weight of the sectional steel cushioned the blow enough to prevent bit failures when 3-in. Leyners were used. Rods used with the 1½-in. bits were ⅞ in. q. o. for sectional steel and 1 in. q. o. for all chuck pieces.

In May 1948, 2-in. tungsten carbide bits became available and test work was immediately started. The 2-in. hole approximated the AX (1$^{15}/_{16}$ in.) diamond drill hole which was being used exclusively for blastholes and permitted their substitution for diamond drill holes in a ring without alteration of pattern, burden, or explosives. The 2-in. bit also gave room in the hole for larger couplings and permitted the use of heavier rods and 3½-in. machines, increasing the

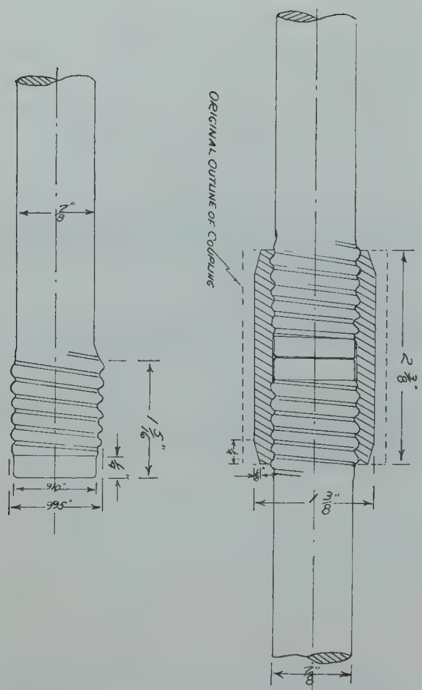

FIG 1—Altered type H coupling and thread.

depth of the hole which could be drilled. The type 2 coupling and 1⅛ in. round rod was adopted for use with this equipment (see Fig 2 for drawing of coupling and threads). Holes up to 55 ft are being drilled, however most of the holes are less than 40 ft.

Drilling Procedures

Drilling is done from 7 by 7 ft blasthole drifts located on the foot and hanging walls of the ore body. The holes are drilled at right angles to the drift and vary from horizontal to minus 90°.

The machine is mounted on a standard 3-in. column with side arm. They are equipped with sliding cone shells so they may be adjusted for required distance from the drilling face as each hole is drilled.

The drills have 30 in. change feeds. This allows two changes of chuck steel before a 5-ft section of additional steel is added. A 7½ ft sectional steel has been used where clearance is available which reduces the number of couplings required, increases drilling efficiency, and reduces coupling cost.

Clearing of cuttings from the holes presented a problem with the 2 in. bit. To overcome this a water tube attached to a ⅜ in. air hose is placed in the end of the steel which causes a continuous blow, forcing the cuttings away from the bit and couplings when pulling rods or placing the bit on the bottom of the hole. It was found also that machines equipped with valves allowing continuous blow of air through the water tube and drill steel to the bottom of the hole gave an air lift action and improved removal of cuttings from the hole.

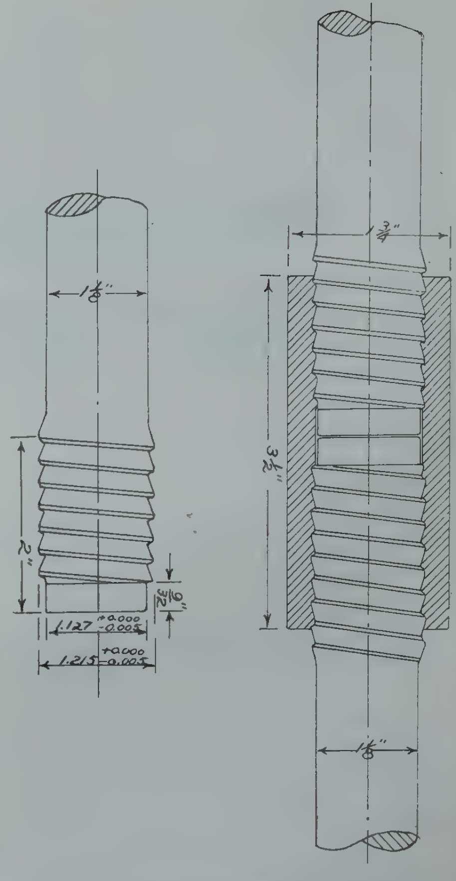

FIG 2—Type 2 coupling and thread.

All holes are started with either 2½ in. detachable bits or conventional steel to prevent undue strain on tungsten carbide bit inserts encountered when collaring a hole on an uneven face. Holes collared in the floor of the drift or in places where the cuttings do not drain away require casings. For this purpose reclaimed 2 in. id pipe is satisfactory.

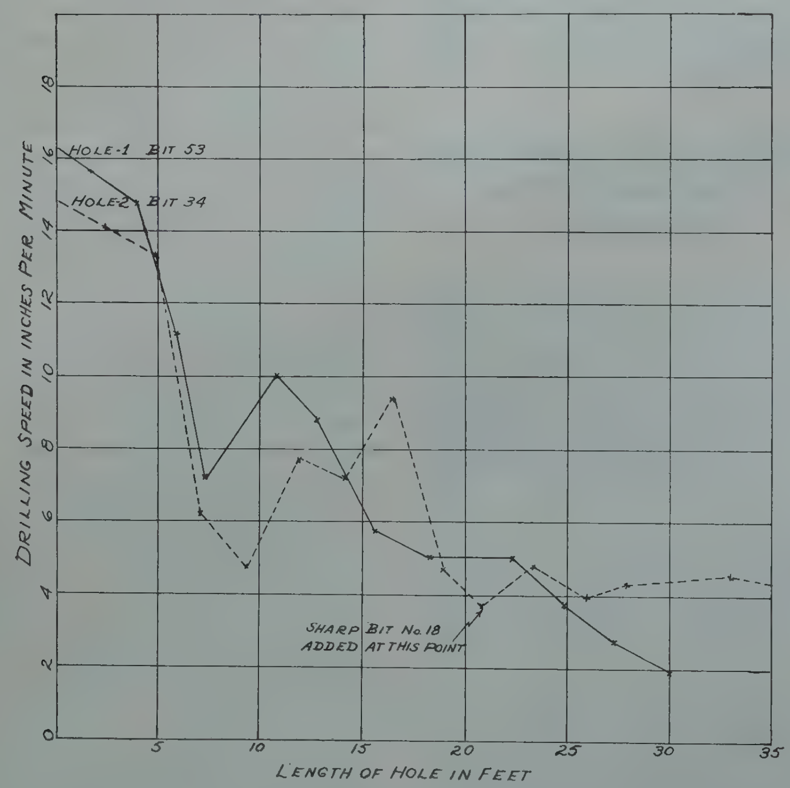

FIG 3—Drilling speeds of 2 in. tungsten carbide bits with sectional steel in holes up to 35 ft in depth.

Drilling Performance

Drilling performance varies considerably from hole to hole. The charts in Fig 3 show the penetration rate in inches per minute for two holes at depths up to 35 ft. Drilling speed varies from 15 to 20 in. per minute at the start of the hole to 2 to 6 in. per minute at 35 ft. The decrease in rate of drilling speed is caused by the additional weight of rod and dulling of the bit. In one hole a sharp bit was

added which showed a marked increase in drilling speed. The variation in drilling speeds was largely the result of changes in physical characteristics of the rocks being drilled. The holes were crossing the formation, passing through changing degrees of silicification and mineralization.

Substitution of Percussion Drill Holes for Diamond Drill Holes in Rings

The illustrations in Fig 4 show the layout of blasthole rings now being used combining 2 in. percussion drill holes and AX diamond drill holes. These ring layouts are not altered from the original diamond drill blasthole pattern. Rings in which 1$\frac{1}{2}$ in. holes are drilled must be changed by adding holes in the area in which AX holes are to be replaced to give approximately the same powder factor. The footage required to do this is 1.67 times the AX hole footage replaced. The use of 1$\frac{1}{2}$ in. holes has been discontinued because of the required alteration of ring pattern and need of special powder size.

In the undercuts percussion holes have entirely replaced the diamond drills as the length of hole required is less than 35 ft and can be readily handled by the percussion drill. The layout for 2 in. holes here again is identical to those for AX diamond drill holes.

In AX standard ring patterns, about 35 pct of the holes are less than 40 ft in length and are easily drilled by percussion drills. But as holes from 65 to 70 ft in length must be drilled to complete a ring, some method must be found of extending the percussion drill holes to this desired length. When this is accomplished, the percussion drill will completely replace the diamond drill for blastholes. To accomplish this future plans call for experimentation with 4 in. piston diameter machines, independent rotation, water swivels independent of machines, higher water pressure, and rods of steel tubing.

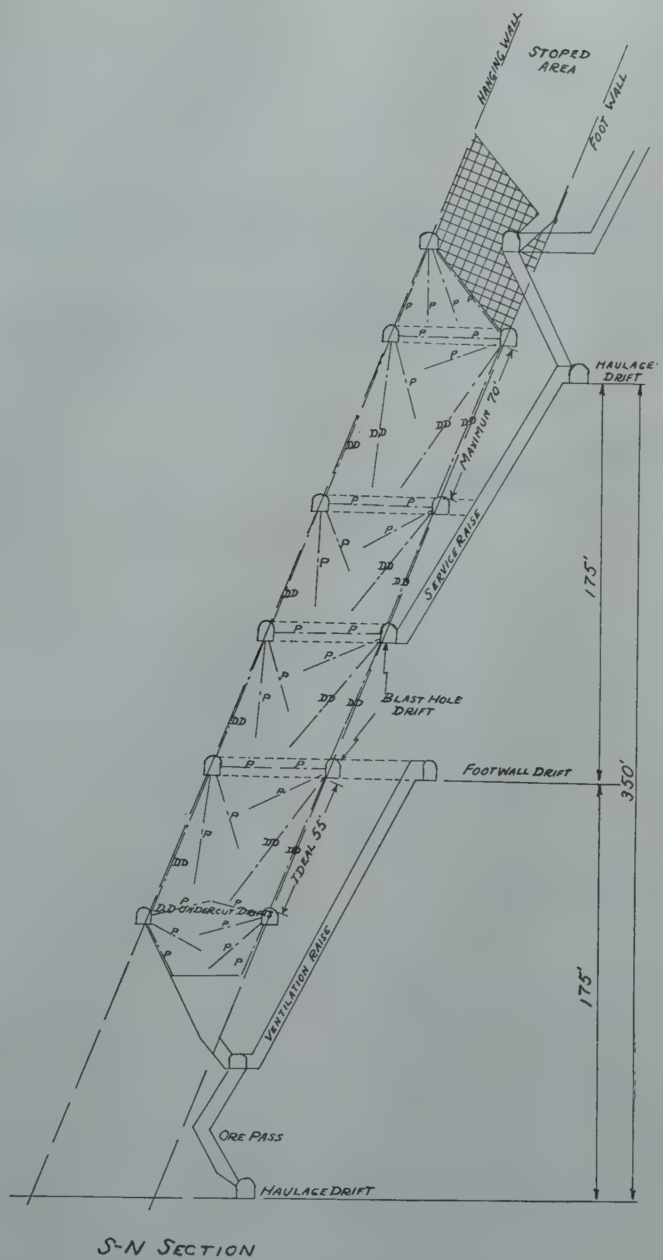

FIG 4—Ideal section showing ring layouts combining 2 in. percussion drill holes and AX diamond drill holes.

Table 1 . . . Blasthole Drilling Data

Period	Number Machines	Total Footage	Number Man Shifts	Number Machine Shifts	Feet Per Man Shift	Feet Per Machine Shift
Diamond Drills, AX Holes						
Sept. A	4	1,219	44	42	27.7	29.0
Sept. B	5	1,846	74	62	22.7	29.7
Oct. A	4	1,564	54.5	51	28.7	30.6
Percussion Drills, 2-in. Holes						
Sept. A	3	1,863	32	24$\frac{3}{8}$	58.2	76.5
Sept. B	3	2,219	50	30$\frac{5}{8}$	44.4	72.4
Oct. A	2	1,728	41	29$\frac{1}{2}$	42.1	58.5

Blasthole Drilling Operations

Output per man shift and machine shift for percussion blasthole drilling is approximately twice that of diamond drilling. This is important as it is difficult to obtain skilled labor to maintain production because of the man power shortage and increased footages per machine drift reduces the investment in equipment required to maintain monthly blasthole footages.

Both diamond and percussion drilling are on a bonus basis. The bonus is based on a minimum footage over which the driller is paid a flat rate per foot. The bonus system resulted in a

substantial increase in output per machine shift for the percussion blasthole machines. Footages drilled per shift vary considerably, but seldom drop to less than 50 ft per machine shift. The maximum footage drilled in a shift was 110 ft in a ring where rock conditions were favorable.

Table 1 summarizes recent diamond and percussion blasthole drill operations.

Costs

The principal consideration when changing from one drilling method to another is one of costs. For comparison purposes percussion and diamond blasthole drilling costs are shown as percentages of total diamond drilling costs in Table 2. This table shows that percussion blasthole drilling costs are 40.2 pct of AX diamond drilling, a saving of 59.8 pct. The largest saving is in tungsten carbide bit costs over diamond bit costs. Tungsten carbide bit costs per foot are approximately one fourth the diamond cost. The next largest saving is in labor which is approximately one half that for diamond drilling.

Two important items appearing under diamond drilling do not show in the percussion drilling cost table. These are supervision and cost of new drills deferred. Supervision of percussion blasthole drilling is under a development shift boss, as part of his regular duties of general mine supervision. No deferred charges for percussion drills are being made as the machines in use have been written off. These two items would increase percussion blasthole costs by about 5 points.

The percussion blasthole drilling cost data shown are for holes up to 40 ft while those for diamond drilling are for holes 65 ft and less. It is expected as the length of percussion drill holes increase the drilling costs will also rise. The saving, however, is still expected to be substantial. Bit cost, which is the item of largest saving, is not expected to increase materially. Some increase in labor, coupling, and rod cost is anticipated.

Table 2 . . . Comparison of Drilling Costs Per Foot

	AX Diamond Drilling Costs, Per Cent	2 In. Tungsten Carbide Bit Percussion Drilling Costs, Per Cent
Drilling labor	23.73	12.62
Bits	54.82	13.27
Drilling accessories (Rods, couplings, etc.)	4.76	10.85
Operating supplies	0.86	0.09
Drill repairs	2.16	3.37
Cost of new drills (Deferred)	6.45	
Supervision	7.31	
Total direct drilling cost	100.00	40.20

Summary

Advantages of percussion blasthole drilling:

1. Drilling
   - *A.* Increased output per man shift.
   - *B.* Increased output per machine shift.
   - *C.* Drillers more plentiful and more easily trained than diamond drillers.
2. Costs
   - *A.* Direct drilling costs are substantially less.
   - *B.* Required investment in equipment is lower.
   - *C.* Bit inventory required is greatly reduced.

Disadvantages of percussion blastholes:

1. Drilling
   - *A.* Length of hole is limited.
   - *B.* Holes have more of a tendency to wander.
2. Loading
   - *A.* Holes are slightly smaller at bottom than at collar.
   - *B.* Walls of holes are uneven.

It is evident that the advantages outweigh the disadvantages and that percussion blasthole drilling with tungsten carbide bits is preferred. Every effort is being made to substitute percussion blasthole drilling for diamond drilling in the future for ore breaking at the Holden mine.

Acknowledgments

The writer wishes to acknowledge the assistance of the engineering and operating personnel of Howe Sound Co., Chelan Division, in preparing this paper. He is also indebted to the representatives of percussion drill and tungsten carbide bit manufactures who aided in development of percussion blasthole drilling at the Holden mine. He also wishes to thank Mr. J. J. Curzon, Manager of the Howe Sound Co., Chelan Division, for permission to publish this paper.

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Sampling of Coal for Float-and-sink Tests

By A. L. BAILEY* and B. A. LANDRY,† Member AIME

Foreword

By THOMAS FRASER‡

All who are even generally aware of the tremendous rate of increase in coal washing operations must realize the growing importance of the float-and-sink test. I believe it is conservative to estimate that, in the past decade, the dollar volume of float-and-sink testing has increased tenfold. It is a simple matter of economy, then, to examine the factors that determine the cost of adequate float-and-sink testing. When the Coal Preparation Section of the Bureau of Mines entered upon a greatly expanded program of such work in connection with the synthetic liquid fuels investigations, it seemed advisable to examine these factors experimentally.

The principal consideration that differentiates float-and-sink test sampling from general purpose sampling, is that the original particle size must be preserved. Therefore, the total cost of the test will be directly affected by any standard that might be proposed to limit sample bulk reduction at any given particle size. For this reason, the relationship of sample size to variability of results was the first factor to be studied experimentally.

Of course, the matter is rendered complex by the circumstance that the float-and-sink test, not a simple analytical measurement but a process test, comprehends a number of more-or-less independent items of fundamental data; and as shown in the report, the variability of the samples differs with respect to these different items.

This condition and the wide variety of situations in which float-and-sink test data are used, in combination with other factors, to study complex process operations, indicate the difficulty of setting up fixed standards for float-and-sink sampling and testing. At this stage at least, it is the intent rather to obtain experimental data on the principal factors involved so that the reader may arrive more intelligently at a procedure adapted to his problem.

The authors of this paper have presented experimental data showing the relationship between size of sample and particle size for different variability tolerances with respect to percentage of sink. In further studies, data are being collected to appraise also the variability of the samples with respect to float-ash content and size consist. The scope of this work will be broadened to cover particle sizes up to 4 in., and a third series of tests has yielded similar data for a much cleaner type of raw coal. Thus, the further studies will make available a fairly comprehensive measure of variability with respect to size consist, percentage of sink, percentage of middlings, and percentage of ash in the float, for three coals ranging from 3.87 to 17.68 pct in refuse content (heavy sink material) and from 4.18 to 17.52 pct in middlings.

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Introduction

At present there are no published standards for float-and-sink test sampling. During the rapid expansion of float-and-sink test work, varying procedures have been based on adaptations of the ASTM standards for sampling coal for analyses. For this reason, a special study of gross sample reduction has been undertaken to determine the limits for this step in the operation where no reduction in particle size is to be made before testing.

Float-and-sink tests are made whenever a thorough study of coal characteristics is desired. The tests may be made on samples from coal-cleaning units such as jigs or tables, or coal samples may be tested which are taken from a loading boom, railroad car, or the coal seam itself. The resultant gross sample may be large and pose a problem of sample reduction. The question is, then, how much can the sample be reduced and still fall within preassigned limits of accuracy of the original gross sample of coal?

Coal from channel samples may be crushed to liberate impurities and then separated into various gravity fractions from which washability curves are drawn; from these curves, it is possible to determine the cleaning characteristics of the coal. However, coal samples from coal-cleaning units cannot be

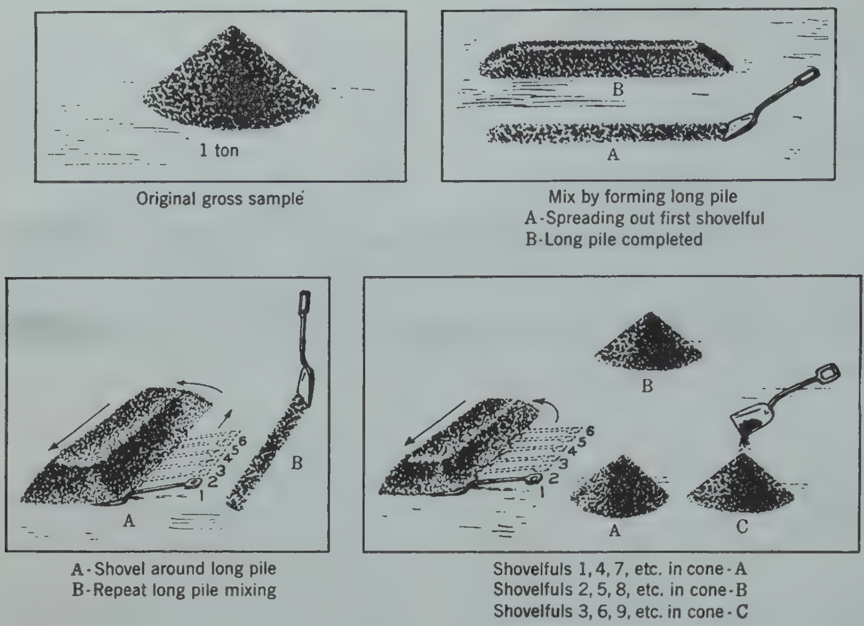

FIG 1—Method of reducing gross sample preliminary to riffling.

crushed without changing the original characteristics of the sample being studied. If the feed or the products are crushed after sampling and the float material is liberated from the sink, a true picture of the cleaning efficiency of the washing unit cannot be determined.

It is necessary thus, actually to test the samples at coarse particle size, and the establishment of minimum tolerances for sample reduction in relation to particle size becomes very important. It is not just a step in the sequence of riffling and crushing operations, but it determines the quantity of material to be handled through the rather laborious and time consuming float-and-sink separations of the washability test.

The work reported herein was undertaken to determine, from tests, the sampling characteristics of a Pennsylvania anthracite and of a Western Pennsylvania bituminous coal to relate these characteristics to the fundamental variables of the coal affecting the variability of the laboratory samples of various weights, and to determine the size of sample required to meet any preassigned degree of representativeness.

Although the results obtained are strictly applicable only to the two coals mentioned, the agreement in the relations established for the two coals suggests that relations of the same kind will be found for other coals. It is believed that extension of the work to only a few more typical coals should yield a good basis for establishing sampling specifications.

It was recognized that the variable most likely to affect the size of sample required to meet a given preassigned accuracy would be the state or degree of mixing of the coal. In order to ensure that the state of mixing would be as favorable as possible, a mixing stage was interposed before each ordinary riffling stage. The complete procedure for obtaining the samples, from which the variability of the float, middlings, and sink fractions were determined, is outlined below. The method described herein represents a rapid and relatively easy means of gross-sample reduction especially suited for field work; it appears that the method can be further simplified in view of the results obtained.

General Information

Two gross samples of raw coal of approximately one ton each were used for these tests. They were a modified 1⅛ in. by 0 Western Pennsylvania high-volatile bituminous coal, with a portion of the ⅛ in. by 0 removed at the mine, and a 13/16 by 9/16-in. Pennsylvania anthracite. Both coals were handled in a similar manner, except that the denser structure of anthracite necessitated the use of a higher-gravity separation.

The work was divided into five phases:

1. Mixing of the coal in long piles followed by riffling.
2. Screening out of undersize.
3. Float-and-sink separation.
4. Particle count.
5. Assembling and interpretation of data.

The gross sample of coal was spread on a smooth, clean concrete floor, then picked up in shovels and each shovelful spread out to form a long, uniform pile as illustrated in Fig 1. This pile was about 30 ft in length. Then by shoveling around this pile, and spreading each shovelful on top of the preceding shovelful, another pile similar to the original pile was made. The pile was then split into three portions by shoveling around the pile—all numbers 1, 4, 7, 10, and so on, shovelfuls being placed in one conical pile, then numbers 2, 5, 8, 11, and so on, being placed in another pile, and numbers 3, 6, 9, 12, and so on, being placed in a third pile. This continued until the long pile had been split into three conical piles of approximately equal size, which were designated piles *A*, *B*, and *C*. Pile *A* was then ready for riffling; the sampler filled his shovel from the base of the cone, walking around the cone for succeeding shovelfuls. These shovelfuls were placed in six riffling pans and riffled as explained below. The same procedure was followed for the other two piles.

RIFFLING PROCEDURE

There were two essential differences between the method of riffling used and that usually practiced. First, a riffle larger than the usual size, with four 2-in. slots in each direction was used. Second, a double riffling cut was employed which was obtained by giving to each pan on the first pass a quarter turn to cut the coal in opposite directions and produce a checkerboard pattern. The pans were riffled through opposite sides of the riffle on the second and final pass to prevent any slight riffle inaccuracy from affecting the split.

The general plan adopted for the study was thus to divide each of these gross samples into a large number of unit portions of approximately 10 lb each and to test all of these unit portions separately to obtain the fundamental data for a statistical study of the precision of sampling; obviously these 10-lb units could be grouped by

computation to obtain data for study of larger units.

Coal did not clog up the slots of the riffle since the slant length of the slots was held to a minimum of about 4 in. long at an angle of 55°, and also since the +1-in. size coal was of a relatively small percentage of the total weight and consequently did not bridge across and blind off the slots of the riffle. This procedure was continued until the total one-third portion had been split into 64 portions designated as laboratory samples. Each sixty-fourth had been handled 12 times—6 mixing cuts and 6 final cuts. The same procedure was followed for piles *B* and *C*. The samples were then placed in bags and numbered *A*-1, *A*-2, *A*-3, and so on, for the first third; *B*-1, *B*-2, and so on, for the second portion; and *C*-1, *C*-2, through *C*-64 for the third portion.

It will be obvious that the above procedure was needed in this particular study because the gross sample was to be entirely retained in a multiplicity of final unit samples. In any routine sampling operation, to reduce a gross sample to one final sample for testing, all the alternate portions would be rejected as they are separated out through the sampling procedure.

SCREENING

It was decided that the finer-size coal particles should be removed from the laboratory samples to facilitate float-and-sink separation which is particularly difficult on the finer mesh sizes of coal. All laboratory samples were hand-screened on an 8-in. diam Tyler sieve and given a total of 20 vibrations with particular care not to overload the screen. The bituminous coal was screened at 4-mesh and the anthracite at 6-mesh.

Float-and-sink Separation

The Pennsylvania bituminous 1⅛-in. by 4-mesh coal was separated at 1.60 and 1.35 sp gr into three fractions. These were a 1.60 sink or refuse separation, a 1.35 by 1.60 middlings product, and a 1.35 float clean coal product. The Pennsylvania anthracite, 1³⁄₁₆ by ⁹⁄₁₆ in., was dedusted on 6-mesh to remove degradation and was separated at 1.85 and 1.60 sp gr, giving a sink, middlings, and float-coal separation. The gravity was changed in this case to fit the characteristics of the coal.

Bromoform, carbon tetrachloride, and gasoline were the organic liquids used to make up the separating mediums. The liquids were combined in various proportions to give the necessary gravity ranges. Organic liquids rather than zinc chloride or some other salt were used, as they leave no residue, require no washing of the coal, evaporate rapidly, penetrate and wet the coal thoroughly, and are not corrosive to the skin. However, the vapors of these mixtures are noxious and anesthetic, and ample precautions were taken to provide adequate ventilation. These vapors are heavier than air, and for this reason, and also because it is necessary to bend over the solution containers, all separations were made with the containers encased in a box equipped with down-draft ventilation.

The float-and-sink containers were made in two pieces, the outer, or solution container holding 7½ gal of solution and the inner container having a removable 20-mesh screen bottom.

The procedure for separation of Pennsylvania bituminous coal was as follows: the sample was first placed in 1.35 sp gr solution and the 1.35 float skimmed off and allowed to dry. The inner container holding the sink product then was immersed in a solution of approximately 1.60 sp gr to rinse off the 1.35 solution still remaining on the material. The container was placed then in the 1.60 sp gr solution and separated into 1.35 by 1.60 sp gr middlings and 1.60 sp gr sink. The Pennsylvania anthracite was separated in a similar manner, with a rinsing solution of approximately 1.85 sp gr also used.

All products of the separations were air-dried for 24 hr, weighed, and their percentages calculated.

Particle Count

From the 192 laboratory samples obtained, 15 samples of approximately average weight were chosen for particle count. The float coal, middlings, and sink fractions were counted for the Pennsylvania anthracite, and the float coal and middlings for the Western Pennsylvania bituminous coal. The sink fraction of this coal had been discarded and could not be counted. The bituminous coal was screened on 1, ¾, and ½-in. round-hole screens and a 4-mesh Tyler sieve and the anthracite on ⁹⁄₁₆ in. and 4-mesh screens. The plus 4-mesh particles were weighed and counted in each case. The number of particles and their weight were averaged for both coals.

Treatment of Data

The primary treatment consisted of calculating the float, middlings, and sink percentages of each of the samples. From these percentages the (standard deviation)² were calculated by the formula

$$\sigma^2 = \frac{\Sigma y_i^2}{N} - \left(\frac{\Sigma y_i}{N}\right)^2$$

where y_i represents the percentage of float (or middlings, or sink) calculated, and N is the number of samples.[1]

As has been previously explained, each gross sample, because it was originally divided into thirds, resulted in a total of 3 times 64, or 192 samples. To obtain the variability of the groups of previous rifflings, an arithmetical recombination of the weights of float coal, middlings, and sink of contiguous samples made it possible to calculate the corresponding σ^2, after converting weights to percentages. These calculations gave the variability that would have resulted if riffling had been stopped after 96, 48, or 24 laboratory samples had been obtained. As could be expected, the samples, when recombined, showed a greater degree of precision because of increased weight.

Table 1 summarizes the variability data obtained for the two coals investigated. The data presented in this table serve as a basis for Fig 3 and 4, and are discussed later in connection with those figures.

Fundamental Variables Affecting Variability of Percentage of Float-and-sink Fractions

In coal sampling, whether it be for the determination of the average ash percentage, or, as in the case with which we are concerned here, for the determination of the average percentage of float material, of middlings, or of sink material, the main problem is to relate the variability of percentages obtained with samples of a given weight to the variables which cause this variability.

In coal sampling for ash, it has been shown[2] that the variability of ash per-

[1] References are at the end of the paper.

centage in increment samples is related to the fundamental variables by the relation:

$$\sigma_W{}^2 = \sigma_{\bar{w}}{}^2 \left(\frac{W}{\bar{w}}\right)^{a-1} \qquad [1]$$

where $\sigma_W{}^2$ is the (standard deviation)^2 of the ash percentage of all increments of weight W in the coal, $\sigma_{\bar{w}}{}^2$ is the (standard deviation)^2 of the ash percentage of the pieces of average weight (weight-weighted basis) in the coal, $\bar{w}$ is this weight-weighted average weight of pieces and $(a - 1)$ represents the degree or state of mixing of the coal.

Table 1 . . . Determined Variability of Float, Middlings, and Sink Fractions

Average Weight of Laboratory Sample, Pound	Float	Middlings	Sink
	(Standard Deviation)^2 σ^2		
Pennsylvania Anthracite, 13/16 by 9/16 Inch			
12.44	1.2462	1.0080	1.3410
24.88	0.7274	0.5843	0.7000
48.76	0.5180	0.3247	0.4073
99.53	0.2893	0.1902	0.1956
Western Pennsylvania Bituminous Coal, 1⅛-Inch by 4 Mesh			
9.63	1.4784	1.0426	0.8133
19.27	0.6724	0.5188	0.4893
38.54	0.4087	0.3749	0.3035
77.07	0.2640	0.2121	0.1775

The concepts and principles that served to establish Eq 1 as the fundamental relation governing the sampling characteristics of coal for ash percentage are also applicable to a coal which is to be sampled for float and sink determinations. The equation may then be written in the form:

$$\sigma_W{}^2 = \sigma_w{}^2 \left(\frac{W}{w}\right)^{a-1} \qquad [2]$$

where $\sigma_W{}^2$ is (standard deviation)^2 of the percentage of float (or middlings, or sink) in laboratory samples of weight W, $\sigma_w{}^2$ is the (standard deviation)^2 of the percentage of float (or middlings, or sink) of the individual pieces in the coal, compounded on a weight basis, w is the average weight of piece, and $(a - 1)$ represents the degree of mixing of the coal with respect to the property under consideration.

Eq 2 may be linearized by taking common logarithms on both sides giving:

$$\log \sigma_W{}^2 = \log \sigma_w{}^2 + (a - 1) \log \frac{W}{w} \qquad [3]$$

which can be plotted as a straight line on log-log paper. Eq 3 is the basis of the log plots presented later.

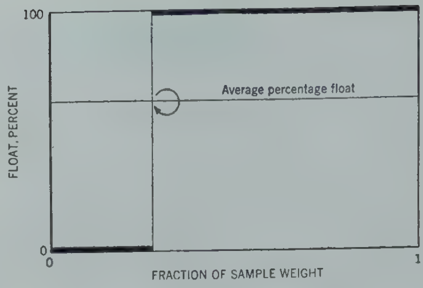

FIG 2—Diagrammatic illustration of Equation 4.

CALCULATION OF $\sigma_w{}^2$

The (standard deviation)^2 of the percentage float, say, of the individual pieces, compounded on the weight basis, is calculated from the expression:

$$\sigma_w{}^2 = \frac{W_F\ (100)^2}{W} - \left(\frac{100\ W_F}{W}\right)^2 \qquad [4]$$

where W_F is the average weight of float material in the laboratory samples and W is the weight of the sample. The same expression is used for the middlings and sink.

Fig 2 has been prepared to illustrate the significance of this calculation. If pieces of coal are selected one at a time, subjected to the float test, segregated in two piles according to whether they float or do not and each pile weighed, a plot similar to that of Fig 2 can be made on which the fraction not floating is plotted on the zero axis and the fraction floating is plotted on the 100-pct float line. The (standard deviation)^2 given by Eq 4 corresponds to the moment of inertia of the two heavy lines with respect to the average percentage of float taken as the axis of rotation.[3]

Table 2 shows the average weight of float, middlings, and sink material, and the average total weight of the laboratory sample, used for calculating $\sigma_w{}^2$ by Eq 4.

Table 2 . . . Average Weight of Laboratory Sample and of Float, Middlings, and Sink Fractions

	Pounds			
	Float	Middlings	Sink	Total
Pennsylvania anthracite	8.06	2.18	2.20	12.44
Western Pennsylvania bituminous	7.42	1.21	1.00	9.63

Table 3 gives the average percentage of float, middlings, and sink of all the individual laboratory samples of the two coals studied.

Table 3 . . . Average Percentage by Weight of Float, Middlings, and Sink Fractions of Individual Laboratory Samples

	Percentage		
	Float	Middlings	Sink
Pennsylvania anthracite	64.80	17.52	17.68
Western Pennsylvania bituminous	77.05	12.57	10.38

Table 4 gives the results of the calculation of the variability of float, middlings, and sink by individual pieces, weight compounded, by means of Eq 4 for use in Eq 3.

Table 4 . . . Variability of Percentages of Float, Middlings, and Sink, Individual Pieces, Weight Compounded

	$\sigma_w{}^2$		
	Float	Middlings	Sink
Pennsylvania anthracite	2,280	1,450	1,460
Western Pennsylvania bituminous	1,770	1,100	930

DETERMINATION OF w

For the determination of the average weight of piece in the laboratory samples, a particle count had to be made. This was done, as already mentioned, for 15 laboratory samples out of the 192 obtained for each coal. Table 5 summarizes the results of dividing the average weight of sample by the average number of particles per sample.

Table 5 . . . Average Weight of Piece in Laboratory Samples

	Pennsylvania Anthracite	Western Pennsylvania Bituminous Coal
Average weight of sample, lb.	12.44	8.63
Average number of particles	3,503	6,186
Average weight of particle, lb.	0.00355	0.00140

DETERMINATION OF COEFFICIENT $(a - 1)$ INDICATING DEGREE OF MIXING

The coefficient $(a - 1)$ in Eq 3 is obtained as a byproduct of the plotting of other known quantities related together by this equation; values obtained will be given below following presentation of the plots of Eq 3 from the results obtained for the two coals investigated.

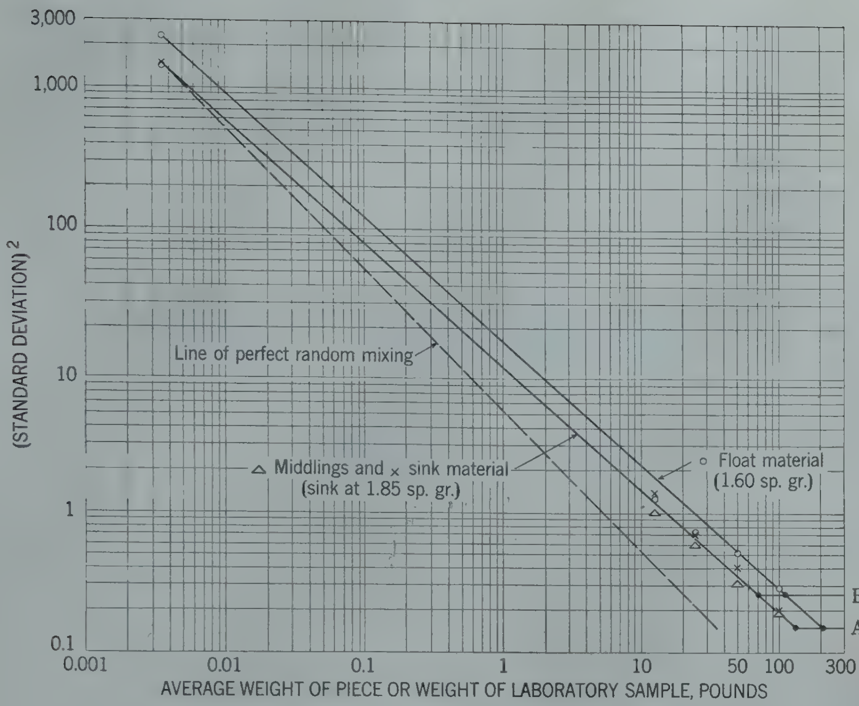

FIG 3—Sampling characteristics of Pennsylvania anthracite. Line *A* refers to an accuracy of ±1 pct, 99 times in 100. Line *B* refers to an accuracy of ±1 pct, 95 times in 100.

Sampling Characteristics of Pennsylvania Anthracite

Fig 3 brings together the results assembled in Tables 1, 4, and 5 relative to the sampling characteristics of the Pennsylvania anthracite investigated. The points at the upper left corner represent the variability of the individual pieces of coal with respect to float, middlings, and sink percentages, from Table 4, plotted against the average weight of piece from Table 5. The individual points plotted in the lower right corner represent the variability or (standard deviation)^2 of float, middlings, and sink percentages in the laboratory samples from Table 1, plotted against the average weight of the laboratory samples. Straight lines have been drawn connecting the upper point to the point corresponding to the laboratory sample of greatest weight. The reasons for plotting the lines so that they pass through this point are discussed later.

The black points at the intersections of the sloping lines with the horizontal lines *A* and *B* give the minimum weight of laboratory sample of this coal that will meet the accuracy associated with lines *A* and *B*. This follows from the fact that as σ_W^2 decreases with increase in the weight of laboratory sample, a value for the variability of samples can eventually be reached, by making W large enough, so that samples of acceptable representativeness will be obtained. The corresponding weight of sample is the minimum weight required.

Line *A* is for an accuracy such that, in the long run, 99 samples in 100 will yield a percentage float (or middlings, or sink) within ±1 pct (of the coal) of the true value, or less.

Line *B* is for an accuracy such that, in the long run, 95 samples in 100 will yield a percentage float (or middlings, or sink) within ±1 pct (of the coal) of the true value, or less.

The (standard deviation)^2 corresponding to lines *A* and *B* are respectively 0.15 and 0.26 and can be computed from normal law probability tables.[4]

Fig 3 shows that for the anthracite investigated and for the method of reduction of the gross sample used, laboratory samples of 210 lb and 112 lb, respectively, will meet the accuracy of lines *A* and *B* for the percentage float determination. Because the sloped lines representing the sampling characteristics of the middlings and of the sink material coincide for this coal, laboratory samples of 130 lb and 70 lb, respectively, will meet the accuracy of lines *A* and *B* for both of these determinations.

Sampling Characteristics of Western Pennsylvania Bituminous Coal

Fig 4 presents, for the Western Pennsylvania bituminous coal investigated, the same type of plot as was used in Fig 3 for the sampling characteristics of a Pennsylvania anthracite. The (standard deviation)^2 and weights corresponding to the various points plotted are given in Tables 1, 4, and 5. The points in the upper left corner represent the calculated variability of the pieces plotted against the average weight of piece. The points in the lower right corner represent the variability of the laboratory samples plotted against the average weight of sample. The straight, sloped lines have been drawn so that they pass through the sample point of greatest weight for reasons to be discussed later.

The black points at the intersections of the sloping lines with lines *A* and *B* give the minimum weight of laboratory samples required to meet the accuracy associated with lines *A* and *B*. These weights are:

1. For the float material, respectively 150 lb and 76 lb.
2. For the middlings, respectively 120 lb and 62 lb.
3. For the sink material, respectively 96 lb and 46 lb.

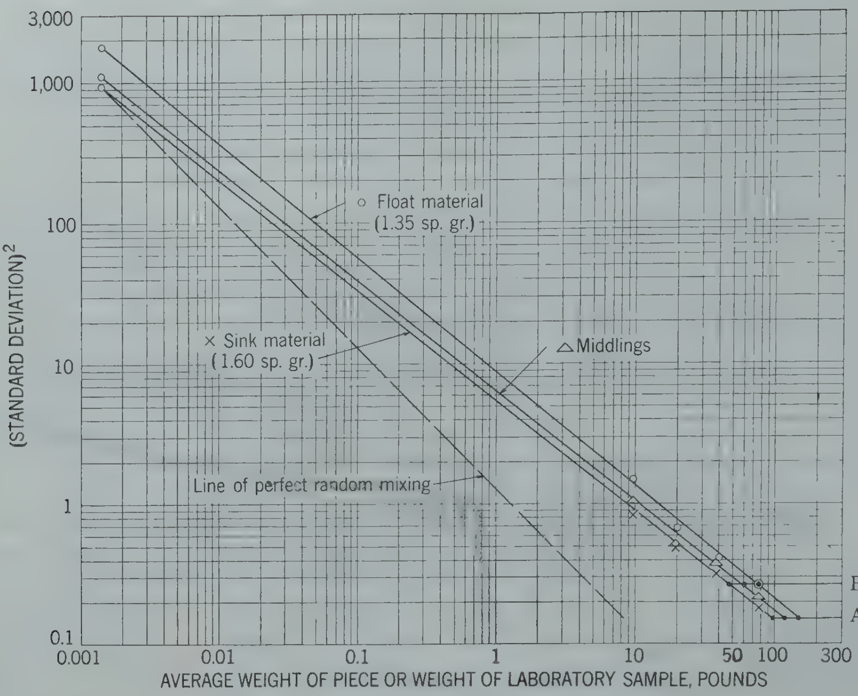

FIG 4—Sampling characteristics of Western Pennsylvania bituminous coal. Line *A* refers to an accuracy of ±1 pct, 99 times in 100. Line *B* refers to an accuracy of ±1 pct, 95 times in 100.

Discussion and Conclusions

As was mentioned previously, the state or degree of mixing of a coal is an important factor in the determination of its sampling characteristics. In the diagram of Fig 3 and 4, the degree of mixing of the two coals sampled is measured by the slope of the inclined lines; these measure approximately −0.88 for the anthracite and −0.78 for the Western Pennsylvania bituminous coal. It can be noted that the inclined lines are closely parallel for the three fractions. The anthracite was slightly better mixed than the bituminous coal.

A slope of zero would indicate perfect segregation. Thus, if the coal placed in the containers before riffling had been so segregated that one container had held all of the sink, another container had all of the middlings, and the remaining containers had all of the float, then the variability of the laboratory samples would have been the same as that calculated for the individual pieces.

On the other hand, if the state of mixing of the original coal plus the mixing operations performed while preparing the long piles and the conical piles, and with the double method of riffling used, had resulted in a coal perfectly random-mixed, then the slope of the inclined lines would have been −1.00 represented by a broken line such as that shown, starting at the calculated point for the sink material, on both Fig 3 and 4. A similar broken line could have been drawn starting at the calculated points for the middlings and float material but is not shown for diagram simplicity.

The rather steep slopes obtained for both coals indicate that the coals sampled were at an advanced stage of mixing, but were not at the complete random-mixed stage.

It is to be noted that in most instances the points representing the variability of the smaller laboratory samples are slightly below the inclined lines passing through the points representing the variability of the laboratory sample of greatest weight. These departures are obviously the result of the added mixing that occurred when the 24 samples were riffled to give the next 48 samples, these riffled to give the next 96 samples, and finally these riffled to give the final 192 samples.

Because the departures from the inclined lines are small and sometimes the departing point is above rather than below the line, it would appear that the added mixing produced by riffling was small and the procedure used of double riffling, at each stage, unrewarding. If, however, the coals had not reached such a high degree of mixing before the double riffling, this method undoubtedly would have resulted in further mixing.

The general agreement that can be observed in Fig 3 and 4 with regard to the lining up of calculated and observed variabilities gives confirmation to Eq 2 and justifies its use in establishing sampling characteristics of coals for determination of float and sink values; and, incidentally, gives added confirmation to the use of the similar Eq 1 in coal sampling for average ash.

It should be emphasized that the weights of laboratory samples that have been found, in this study, to meet the accuracy requirements defined for lines *A* and *B* are a function of the corresponding accuracies. Smaller weights of laboratory samples would have been obtained by making the acceptable accuracy less stringent either by widening the range from ±1 pct of the coal to $\pm y$ ($y > 1$) pct of the coal, or by decreasing the associated probability from 0.95 to $x (x < 0.95)$.

References

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4. Pages 23–24 of ref. 1.

Aerial Photographic Contour Maps for Strip Mines

By GEORGE HESS* and R. H. SWALLOW,† Member AIME

The purpose of presenting this paper is to show:

1. The applications of aerial photography to the map requirements of strip mining.
2. The methods and procedures for producing accurate aerial contour maps.
3. The advantages of aerial contour mapping over stadia or plane table methods.

Introduction

Aerial photography was once a crude, uncertain tool. Today it is a precision mapping instrument which saves important time and money for strip mining and other industry.

Aerial photography began in the boxkite days of aviation. The Army first used it for rough reconnaissance photographs. Then, in the early Twenties, industry began to use aerial photographs of their plants and facilities for advertising or annual reports. The quality of these pictures was not very good by today's standards, but this work began the development of better cameras, better lenses, better photographic techniques.

Then, some farsighted engineers and aerial photographers found that if the camera's axis were vertical to the earth's surface, these pictures could be the source of valuable mapping information. By viewing these pictures in stereoscopic pairs, accurate measurements could be made and contour maps produced from this aerial photographic base. So the aerial mapping business developed, using the plane and aerial camera to collect the data, working with skilled ground parties who established a network of ground control for this map information photographed from the air.

Over the following years, a whole new science developed, known as photogrammetry. Precision cameras and lenses were perfected, and the narrow view of the surveyor's transit was broadened to the vast perspective of the mapping plane.

Stereoscopic Coverage Obtained

Today in making aerial topographic maps, aerial photographs are exposed so that full stereoscopic coverage for the survey area is secured. To obtain true elevations these photographs are processed through a series of precision measuring and contouring instruments which employ a principle similar to the military range finder.

Here is how aerial mapping is being used by your industry. Topographic maps are but one of three products from aerial photographic surveys. *First*, there are the individual photographs. These are taken in flight strips, exposed with an overlap, just as shingles on a roof overlap. Exposures are overlapped from 55 to 60 pct along the line of flight so that a large part of the area in one picture also appears on the adjacent photograph. Successive strips are flown with a side-lap of between 20 and 25 pct. Each point on the ground thus will be somewhere near the center of at least one photograph in the series. They may be studied under a stereoscope and relief seen in a third dimensional effect. Geologic features that may go unobserved in ground exploration are often detected under stereoscopic inspection. Moreover, though field checking is not obviated with these aerial photographs, it is directed toward critical areas and toward worthwhile anomalies.

Aerial Photographic Mosaics

The *second* product of an aerial survey is the photographic mosaic. This is an assembly of carefully matched aerial photographs in mosaic form. Although not as accurate as the topographic map, it does reveal a wealth of cultural detail in true relative scale. Since the mapping plane is flown over the area at a fixed altitude, the peaks and valleys are pictured at different scales. During compilation these scale differences are compensated by adjusting one vertical picture to another according to plotted ground control positions. Only extreme variations in ground elevation cannot be

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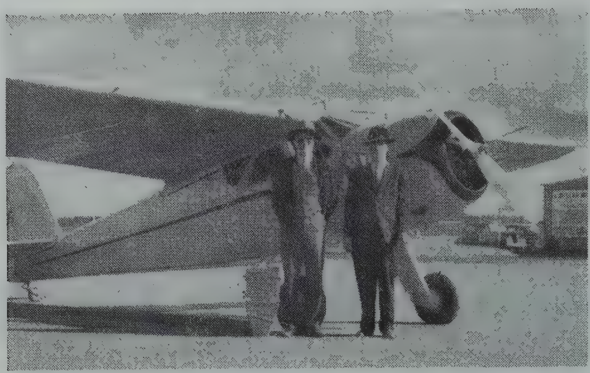

FIG 1—Pilot and photographer mapping team.

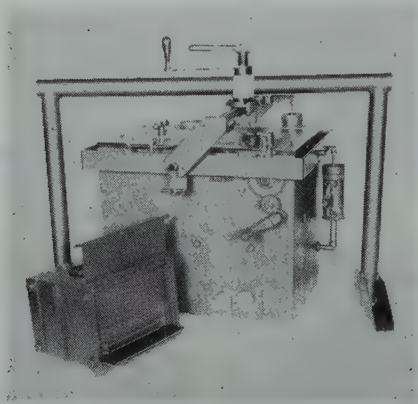

FIG 2—The Brock camera, which employs selected flat glass plates for precision mapping.

FIG 3—Field survey crews establish minimum ground control.

reconciled accurately. In reasonably flat country, the mosaic is a good map for any use except where contours are required.

Finally, the most refined product of the aerial survey is the topographic map. These conform to any specifications set up for mapping performed by ground methods. It is far faster than ground methods—its cost compares favorably—and it is more accurate.

Aerial Method Appraised

You are familiar with plane table and stadia methods. The illustrations and following explanation may help you to appraise the aerial method: first, the mapping team—pilot and photographer (Fig 1). They must work as a highly coordinated team because the mapping plane moves along at about 120 miles an hour, and pictures are made at intervals of 10 to 20 sec. There is no time for discussion or argument. The pilot must develop a high degree of skill in keeping the plane exactly on course and parallel to the earth's surface; the photographer must operate his precise mapping camera rapidly, checking the flight course for full coverage.

Mapping flights are made in late fall, winter, or early spring because then foliage does not obscure the ground. Most mapping photographs are taken between 10 a.m. and 2 p.m. for best lighting conditions. The plane used is often a single-engine craft—a light, stable, workhorse plane.

The Brock camera used for topographic mapping is an important refinement over the film type cameras. In the past, aerial pictures usually have been taken on film which is subject to expansion and contraction in developing and processing. This leads to significant distortions which no process can fully correct. The Brock camera, however, uses selected flat glass plates because of their dimensional stability (Fig 2).

Minimum Ground Control

In constructing the topographic map from these aerial photographs, a minimum amount of ground control is needed. Base control is run with theodolites, carrying horizontal and vertical measurements simultaneously. Horizontal closures average better than 1 in 20,000 and vertical control must close 0.05 ft per mile run.

Radio transceivers are used for communication between instrument men and rodmen. Since most shots exceed 3000 ft, visual signals are not practical (Fig 3).

Tilt Correction and Contouring Accomplished

Truly vertical photographs with the plane of the picture absolutely horizontal are exceptional, for rarely can an airplane be kept in alignment for a series of exposures. Thus minor tilts occur when the pictures are taken. Correcting projectors eliminate the tilt that occurs in the original photography; they are calibrated to correct for as little as 30-sec tilt (Fig 4). Even this, if left uncorrected, would affect the accuracy of the finished map.

After such correction, the plates are placed in a measuring stereometer, which operates on the same principle as the old parlor stereopticon (Fig 5). However, this has been refined and perfected so that the operator sees a brilliant third-dimensional model. Within the optics of the stereometer are cross-hairs acting as a floating mark, which the operator easily and accurately brings to co-incidence with the ground. A setting is made for each contour, and the operator traces all points at this elevation onto the transparent overlay which covers the stereoscopic image. He works through the successive elevations, until the entire area has been contoured according to the predetermined scale. On work of special urgency, this and the other steps of map compilation can proceed in laboratory and drafting rooms on a double shift basis.

Finally, the individual contoured templates or overlays are assembled and the finished map tracing made (Fig 6).

Scales of maps made by the aerial method may range from 1 in. equals 100 ft to 1 in. equals 1000 ft, and the contour interval may vary from 2 to 5 to 25 ft or greater, as required. Standard specifications provide that 90 pct of the map be accurate within one-half contour interval, and that no part shall be in error by more than one contour interval. The completed maps can be made on any type or size of tracing cloth or hard copy paper.

The topographic map affords an excellent base for the plotting of such details as property owners' names, drill holes, legends, outcrop, and so on.

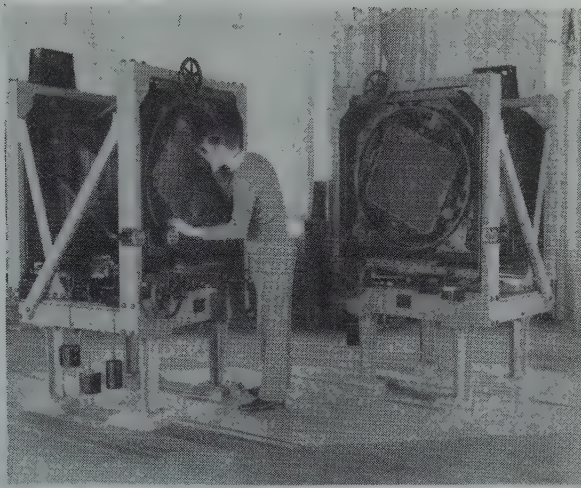

FIG 4—Massive correction projectors eliminate any tilt in the original photography.

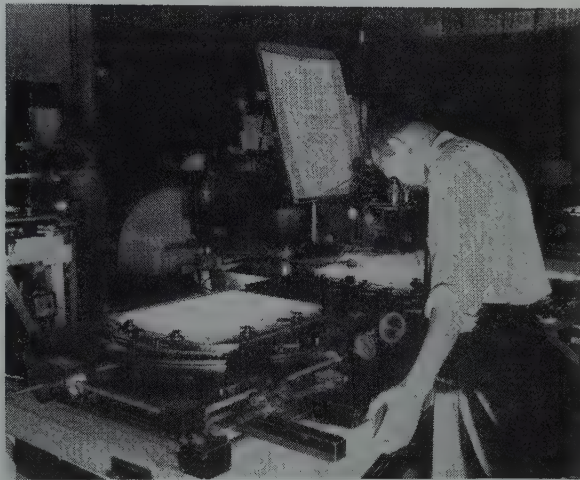

FIG 5—Operator contouring on transparent overlay on stereometer.

Advantages of Aerial Contour Maps

It is not the purport of this paper to explore the uses and advantages of contour maps, for these are well known to open pit mining engineers. However, it is important to consider these advantages of aerial contour maps:

COSTS

Aerial contour maps range in cost from $1.50 to $2.00 per acre. Here is a comparison of costs with maps made by ground methods. Recently one large strip mining company surveyed, traced, and completed maps by the older methods, for an area of 2045 acres. Scale was 1 in. equals 100 ft; the contour interval was 5 ft. Total cost for engineers, draftsmen, helpers, and so on was $5994—or a cost of $2.92 per acre.

At about the same time, a tract with comparable topography was contoured, traced, and completed by aerial methods, at the same scale and contour interval. This area comprised 7210 acres. Total costs for flying, photographing, ground control, and tracing were $13,289—or $1.84 per acre.

While cost will vary according to size of area, topography, and delivery schedule, the aerial method brings a material saving over ground costs on areas of 1000 acres or more. However, on any survey that can be accomplished in less than two months time by one plane table party the advantages of the aerial method are considerably less.

TIME

Important savings in time are

FIG 6—Typical section of typographic map prepared from an aerial photographic base.

possible through the aerial method. The aerial photography requires days; ground methods need weeks and months. Since the aerial method needs only a minimum of ground control it is not necessary to cut through brush or rough terrain. The line of least resistance can be followed because points are not required as frequently as in stadia or plane table methods. Moreover, the drawing of contours is considerably more rapid with the use of stereometers than plane tabling or interpolating stadia field notes. Consequently, the time schedule for production of the finished map is cut at least in half in comparison with the time for the older methods.

GREATER DETAIL

The aerial topographic map is a more refined product; all details, such as ponds, buildings, power lines, fence lines are secured. In ordinary ground methods many of these are missed as a result of human error. Better expression in the actual contour lines is obtained through the aerial method, and no minor terrain change is overlooked.

INDEPENDENT OF TERRAIN

In the mapping of irregular, difficult or inaccessible terrain, the advantages of the aerial method are obvious. The mapping plane soars swiftly over rugged country, independent of the terrain problems facing the ground surveyor.

SUPPLEMENTARY INFORMATION

From the basic aerial photographs made for the contour maps, individual prints and mosaics may be made at a cost of about 4¢ to 6¢ per acre. In prospecting an area where local or government quadrangle maps are not available, these prints or mosaics facilitate reconnaissance.

In flying for contour maps, it is advantageous to overfly the area to be contoured in order to get the boundary features. The contact prints and mosaics, therefore, will cover a larger area than the contour maps. After prospecting and drilling it is then only necessary to have that part of the area contoured which is essential to the future mining operation.

It is only in the last few years that the advantages of aerial contour mapping have been realized. But the aerial method has now proved itself as to accuracy, and as to savings in time and costs. Aerial photography works for your industry today in many areas in West Virginia, Kentucky and Indiana—as well as in the iron ore areas: the Marquette Range in Michigan, and the Mesabi and Cayuna Ranges in Minnesota. It is one of the important new tools which reduces costs and speeds development schedules for the open pit mining industry.

Drying Low-rank Coals in the Entrained and Fluidized State

By V. F. PARRY,* J. B. GOODMAN,† and E. O. WAGNER†

Summary

The low-rank coals containing 10 to 50 pct natural bed moisture represent over half of the tonnage reserve of the available solid fuels of the United States, but only about 2 pct of United States coal production is derived from these fuels. Increased utilization of these large reserves will result if the bed moisture can be removed at the mine in high-speed low-cost processes. This paper presents a theoretical and experimental study of the factors affecting high-speed drying of coal dusts.

The time required to heat particles of coal is proportional to the square of their diameter, and it is calculated that the probable time in minutes is equal to $6D^2$. In other words, a ⅟₁₆ in. coal particle should be heated to drying temperature in 1.5 to 2.0 sec. Likewise, ⅛ in. particles should be heated in about 6 sec. The net heat required to dry subbituminous coal and lignite varies from 240 to 375 Btu per pound, depending upon the moisture removed, and this heat can be obtained from 42 to 61 cu ft of hot gases at 600°F. The necessary heat also can be obtained from 7.5 to 11 cu ft of gas at 2000°F.

A description of the design and operation of a pilot plant for flash drying coal dusts is presented. This plant dries ⅟₁₆ in. by 0 coal dust in suspension in 1.5 to 2 sec with 600°F gas at a rate of 1300 lb per hour per square foot of column area and operates continuously at an efficiency of about 85 pct. The design and operation of another pilot plant for drying ⅛ in. by 0 coal dusts is also described. This unit dries coal in the fluidized state using the sensible heat of gases introduced at 2000°F and processes coal at a rate of about 1500 lb per hour per square foot of grate area at 85 to 90 pct efficiency.

Over 90 pct of the bed moisture of low-rank coal dusts can be removed by the methods described. The heating value of lignite can be increased 45 pct and that of subbituminous coal 25 pct in a few seconds. The physical and chemical properties of coal dusts before and after drying are presented in the report, and it is shown that the bulk density of the dried coal is equal to that of the raw coal. This phenomenon makes it possible to ship 25 to 45 pct more heat in a railroad car with consequent savings in freight costs.

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[1] References are at the end of the paper.

Introduction

For several years the Subbituminous Coal and Lignite Section of the Coal Branch of the Bureau of Mines at Golden, Colo., has been concerned with upgrading low-rank coals by removal of natural bed moisture. These fuels contain 10 to 50 pct total moisture, which reduces their industrial value considerably. In 1938, investigations were initiated in cooperation with the University of North Dakota to study drying of lignite by the Fleissner process with heat derived from high-pressure saturated steam. Two pilot plants were built, and two reports were published giving the results of this work.[1,2] It was demonstrated that lump lignite and subbituminous coal

could be dried readily by the Fleissner process, but it was not feasible to treat slack sizes. From pilot-plant tests it was proved that the capacity of steam-drying plants treating 1 by 2 in. lump should be in the range of 100 lb per hour per cubic foot of autoclave volume; and it was indicated from studies of costs of large plants that the cost of drying lignite lump should be 35 to 50 cents per ton based on 1940 price levels. Experimental work was carried on intermittently for several years, but no commercial plants were built because coal producers and industrial users desired to dry slack coal in most instances, and the upgrading of lump lignite was not particularly attractive to them.

The removal of bed moisture from the slack sizes of lignite and subbituminous coal is somewhat more difficult than the removal of surface moisture from ordinary bituminous slack coal, and the methods used for the latter have not been very successful with the low-rank coals because of the additional time required to remove moisture from the interior of the coal lumps. The rotary drier does not perform efficiently on subbituminous coal or lignite, and considerable dust is lost in the processing. For example, one plant treating ¾ in. slack coal at a rate of 10 tons per hour reduces the total moisture content from 31 pct to only 20 pct in 30 to 50 min through a 50 ft kiln. Another plant treating ¾ in. slack at a rate of 12 tons per hour in tandem kilns totaling 90 ft in length reduced the total moisture from 12 to only 6 pct in about 90 min. The efficiency of these kilns, operating on subbituminous coals, is only about 43 pct, and the loss of dust is considerable. The foregoing plants are the only commercial units now operating on low-rank coals in the United States, but much interest has been expressed by operators in the possibilities of upgrading these fuels by drying, particularly for preparation preliminary to briquetting.

The growing interest in utilizing the low-rank fuels as raw material for synthetic fuels and for other industrial purposes has focused attention on cheap, high-speed methods for removing bed moisture. If these fuels can be dried cheaply at the mines and transported and stored at industrial plants, the present barriers to utilization would be largely removed because the heating value of dry lignite is only 15 to 20 pct less than the heating value of the best dry bituminous coals, whereas, in its natural state, lignite contains only half the heat of good bituminous coal.

Since the natural moisture in low-rank coals is distributed more or less uniformly within the coal substance, the center of each lump must be heated to at least 212°F to remove a substantial portion of the moisture. The report discusses the time required to heat particles of coal, and it is shown that the time is proportional to the square of the diameter. Particles of coal of 14-mesh size (0.046 in.) can be heated in about 1 sec, but a 1.0 in. lump of coal will require about 360 sec. This great difference in time explains why it is inefficient to dry slack coal that varies considerably in size and contains a substantial portion of lump pieces. In other words, slack coal must be subjected to heat for the time necessary to treat the large lumps in order to attain drying. This results in overdrying of the fines, which could be processed in a much shorter time. In a rotary kiln the fine material covers the coarse particles and prevents heat from reaching the lumps. This results in a much longer drying time. As previously mentioned, 1 to 2 hr may be required to process the low-rank slack coals in rotary kilns, while it is theoretically possible to dry 1 in. slack in 6 min if heat can be transferred to the coal.

When bed moisture is removed from the low-rank coals, considerable shrinkage occurs, which makes the dried coal friable and causes degradation during handling. Since the natural tendency of low-rank fuels is to degrade to small sizes, it is believed that they should be processed in the dust form for the large-scale industrial uses.

SUMMARY OF RESULTS

This report presents results of experiments on the flash drying of $\frac{1}{16}$ in. by 0 dusts using waste heat and experiments on drying ⅛ in. by 0 dusts in the fluidized state using high-temperature inert gases. The object of the investigation was to develop engineering data that could be used for designing low-cost, high-speed processes for treating these fuels. The experimental work was done in continuous pilot plants of 100 to 300 lb per hour capacity. The study has been under way for about one year, and this is the first report of progress.

The investigation thus far indicates that waste heat at 600°F can be used efficiently to dry $\frac{1}{16}$ in. by 0 coals in suspension in 1.5 sec, and enough engineering data are available for design of larger plants. If high-temperature gases using primary heat are employed, low-rank coals of ⅛ in. by 0 size can be dried in fluidized beds at very high efficiency and at rates of ¾ to 1.0 ton per hour per square foot of column area. At this capacity, a drying unit for a 2000 ton per day unit would be only 12 ft in diameter and about 15 ft high, and the cost of drying would be in the range of 25 to 35 cents per ton.

The experimental results on fluidized drying indicate efficiencies of 86.5 pct, excluding radiation, and overall efficiencies of 76.4 pct. When operating a large plant, it is calculated that overall efficiencies of 85 to 90 pct can be attained. The space velocity of hot gases in the fluidized column is about 9 fps, somewhat higher than the space velocity of a stable fluid bed, and the density of the fluid bed is 10 to 12 lb per cubic foot.

TIME REQUIRED TO HEAT PARTICLES OF COAL

From theoretical considerations of the time required to heat different sizes of coal to a given temperature distribution, it is evident that the size of coal determines the time required to heat.

The theory of conduction of heat through homogeneous solids, developed mathematically by Fourier and his followers, is not directly applicable to conduction of heat through subbituminous coal and lignite, as these fuels are not homogeneous in structure. In addition, they constantly undergo both physical and possibly some chemical changes during the process of heating to temperatures required to remove all the moisture. Nevertheless, Fourier's mathematical procedure, with suitable limitations and modifications, can be used to calculate the temperature history of particles of coal that are heated in a fluid medium. Burke, Schumann, and Parry[3] have suggested a solution to this problem and have proved that the law of squares can be applied to this case, even though the thermal constants may vary with temperature and the temperature may vary with time. This law states that the time required to heat a body to a definite condition or temperature distribution is proportional to the square of some linear dimension of the body. In the case of spheres, the time required is proportional to the square of the diameter.

When the source of heat is relatively unlimited, such as with condensing

steam or the condition in a fluidized bed, and the properties of the material being heated govern the rate of acceptance of heat, the time required to heat spheres to bring the center temperature to about 95 pct of the surface temperature is:

Time of heating, in minutes

$$= 0.37R^2/K$$

(where K = thermal diffusivity*)[4]
(Refer to Fig 5 of the solution previously discussed[3].)

Consider a solid sphere of subbituminous coal or lignite immersed in an envelope of steam or in a heated fluidized bed. Assume that the outer surface of the lump is heated instantaneously to the temperature of the heating medium and maintained at this temperature. Now if it is assumed that the coal particles have an average specific heat of 0.50, a density of 78 lb per cubic foot and a thermal conductivity of 0.25 Btu per hour per square foot per degree F per foot, the time required to heat the particle to any average temperature distribution can be calculated from the laws of conduction referred to above. Using these constants, the thermal diffusivity of average low-rank coals can be calculated. This term is the ratio of conductivity to the product of density times specific heat, or $K = k/\rho c_p$. It is obtained from the expression giving the general differential equation for unsteady heat conduction derived from the basic Fourier equation for conduction of heat by assuming that conductivity is constant. Although this simplifying assumption is made, it has been proved that the law of squares still applies when lumps of coal are heated.

$$K = \frac{0.25}{78 \times 0.50} = 0.00641 \text{ sq ft per hr.}$$
$$= 0.0154 \text{ sq in. per min.}$$

Heating time, in minutes =

$$\frac{0.37R^2}{0.0154} = 24R^2$$

Heating time, in minutes =

$6D^2$ (D = diameter of particle, inches)

Fig 1 shows a plot of the foregoing equation and expresses the theoretical time required to heat particles of coal in an atmosphere of saturated steam or in a fluidized bed. In simple language it tells us that particles of 0.05 in. diam (14 mesh) should be heated in 1.0 sec, while particles 1.0 in. in diameter should require about 360 sec. The points on this figure are experimental data which will be discussed later. Point A represents the time of contact of 1/16 in. by 0 particles in the flash drier; point B the time of contact of 1/8 in. by 0 particles in the fluidized drier; point C the time required to heat 1¼ in. lumps with saturated steam; and point D the time for heating 1.75 in. lumps in saturated steam.

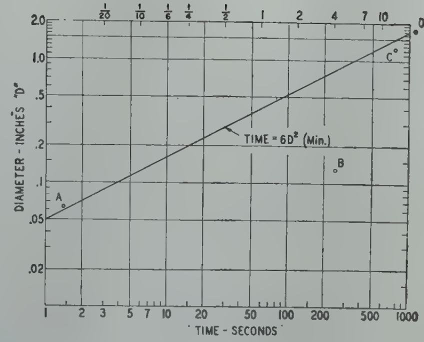

FIG 1—Time required to heat particles of coal.

NOMENCLATURE AND DEFINITIONS RELATED TO DRYING LOW-RANK COALS

Because of the considerable change in weight when low-rank coals are dried, ordinary methods of expressing the moisture removed may be confusing. For example, if a lignite containing 35 pct moisture is dried to a product containing 5 pct moisture, it is commonly considered, by making a simple subtraction, that the coal has lost 30 pct in weight. This is not true, and the estimation is in error by 5 pct. The true weight loss is 31.6 pct. These differences become significant when drying the low-rank coals, therefore it is necessary to adopt a precise nomenclature for expressing results of drying.

The "improvement ratio" = R is the ratio of the weight of coal before drying to the weight after drying. If no dust or fixed gas is lost, this also expresses the ratio of heating value, volatile matter, fixed carbon, and ash after and before drying. The improvement ratio obtained by weight (R_w) is related also to the moisture content before and after drying, and, if no dust or volatile matter is lost, the moisture contents before and after drying must be compatible with the moistures estimated from the weight ratio. It follows also that, if moisture determinations are correct, the dust lost in the system can be calculated by comparing with the weight ratio. Because of the confusion that can arise from inaccurate expressions of results of drying, the authors have adopted the following nomenclature to express results.

Nomenclature

M_1 = percentage moisture in coal before drying.
M_2 = percentage moisture in coal after drying.
W_1 = weight of coal before drying.
W_2 = weight of coal after drying.

$$R_w = \text{improvement ratio} = W_1/W_2 \quad [1]$$

$$R_m = \text{improvement ratio} = (100 - M_2)/(100 - M_1) \quad [2]$$

$$R_v = \text{improvement ratio} = \frac{\text{volatile content after drying}}{\text{volatile content before drying}} \quad [3]$$

$$R_c = \text{improvement ratio} = \frac{\text{fixed carbon after drying}}{\text{fixed carbon before drying}} \quad [4]$$

$$R_a = \text{improvement ratio} = \frac{\text{ash content after drying}}{\text{ash content before drying}} \quad [5]$$

$$R_h = \text{improvement ratio} = \frac{\text{heating value after drying}}{\text{heating value before drying}} \quad [6]$$

$$R = R_w = R_m = R_v = R_c = R_a = R_h \text{ when no dust or fixed gas is lost.} \quad [7]$$

$$U = \text{"ultimate improvement ratio"} = 100/(100 - M_1) \quad [8]$$

This represents the maximum improvement by extracting all moisture without liberation of fixed gases.

$$D = \text{per cent of drying} = \frac{U(R-1)100}{R(U-1)} = \left(1 - \frac{M_2}{R_m M_1}\right) 100 \quad [9]$$

This represents the percentage of total water in the raw coal removed during the drying process.

$$M_2 = 100 - R(100 - M_1) \quad [10]$$

$$W_2 = W_1/R \quad [11]$$

$$L = \text{dust loss, percentage of dried coal} = \left(1 - \frac{R_m}{R_w}\right) 100 \quad [12]$$

$$X = \text{pounds of moisture removed per pound of original coal} = 1 - (1/R) \quad [13]$$

* Thermal diffusivity = k/cs = the rise in temperature produced in 1 cc of the substance by 1 cal flowing in 1 sec through 1 sq cm of a layer 1 cm thick having a difference of 1°C between the faces.
c = Specific heat of material.
s = Specific gravity of material.
k = Thermal conductivity.

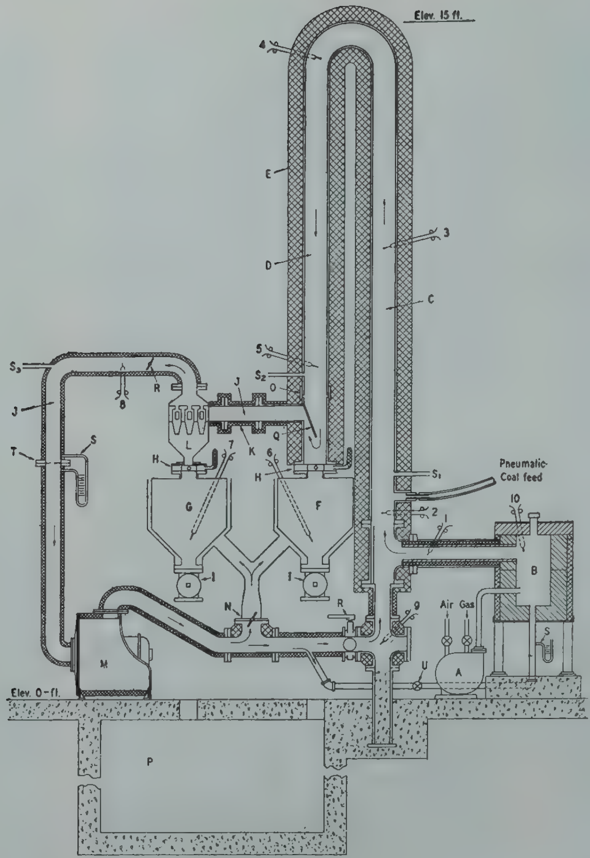

FIG 2—Flash dryer for fine coal.

A "Maxon" air-gas premixer
B Hot gas generator
C Flash drying column—"up section"
D Flash drying column—"down section"
E Mineral wool pipe insulation—3½ in. thick
F Coarse dust receiver—3.45 cu ft capacity
G Fine dust receiver—3.45 cu ft capacity
H Wafer valves—4 and 6 in. and adapters
I Discharge valves—Plug—R.P.—3 in.
J Thin wall ducting—4 in.
K 85 pct magnesia insulation—1⅛ in. thick
L "Aerotec" dust collector
M "Spencer" turbo-compressor
N Waste-gas butterfly valve—4 in.
O Packed slip expansion joint—6 in.
P Dry coal storage pit—5 by 5 by 5 ft.
Q Coarse dust separator
R Regulating valves for recirculation
S Differential and static pressures (S_{1-2-3})
T Recirculating gas orifice meter
U P.O.C. regulating valve

HOT GAS REQUIRED TO DRY LOW-RANK COALS

Since it is necessary to heat the coal particles to at least 212°F to remove a substantial portion of the total moisture, the hot gases must furnish heat at a temperature level higher than this temperature. Table 1 shows the calculation of heat and minimum hot gas required to dry average subbituminous coal and lignite. In this example it is assumed that 95 pct of the total moisture is removed with hot gases at 600 and 2000°F, delivering heat above a temperature level of 250°F. No radiation is considered, and the calculation gives the ideal or optimum conditions of drying. The calculations show that

Table 1 . . . Calculation of Heat and Hot Gas Required to Dry Subbituminous Coal and Lignite

	Subbituminous	Lignite
Total water removed, pct = D	95	95
Total moisture in raw coal, pct = M_1	24	37
Total moisture in dried coal, pct = M_2	1.6	2.8
Dried coal per lb of raw coal = W_2	0.772	0.648
Improvement ratio = R	1.294	1.542
Moisture removed per lb of raw coal = X	0.228	0.352
Temperature of coal charged to drier, °F	60	60
Temperature of dried coal, °F	220	220
Temperature of moisture vapor from coal, °F	250	250
Latent heat in water vapor Btu per lb of raw coal = 1058 X	241	373
Sensible heat in water vapor Btu per lb of raw coal = 84.2 X	19	30
Heat in dried coal, Btu per lb of raw coal at 0.25 sp. ht. = $W_2(220 - 60)0.25$	31	26
Net heat required to dry coal, Btu per lb	291	429
Sensible heat content of average products of combustion between 250 and 600°F, Btu per cu ft	7	7
600°F gas required to dry coal, cu ft per lb[a]	42	61
Sensible heat content of average products of combustion between 250° and 2000°F, Btu per cu ft	39	39
2000°F gas required to dry coal, cu ft per lb	7.5	11.0

[a] Gas volumes expressed at base of 60°F and30 in. Hg.

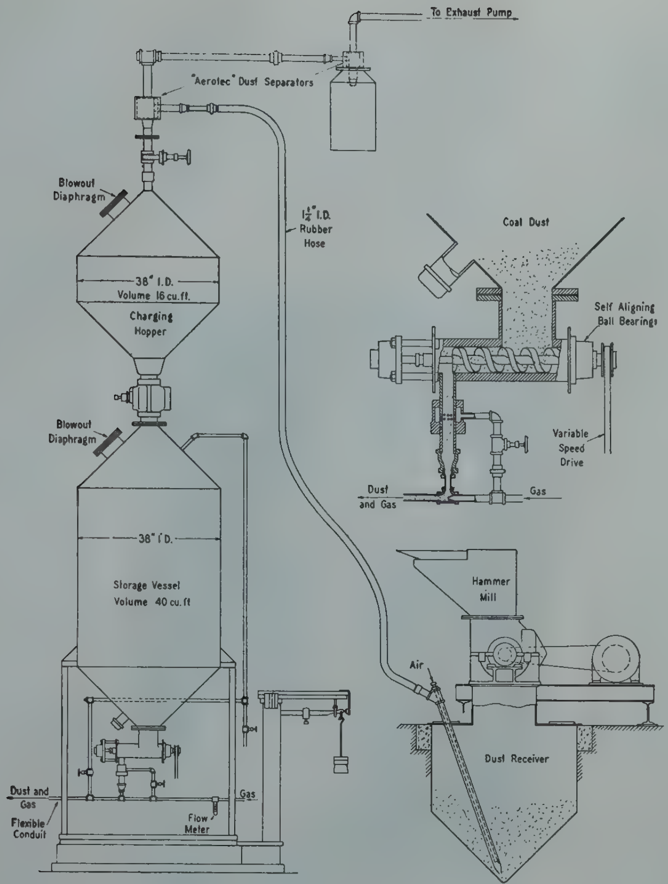

FIG 3—System for preparation, storage, and metering of coal dust.

FIG 4—Flash-drying pilot plants, Golden Colo. The structure on right houses flash-drying column for ¹⁄₁₆ in. by 0 coal, and the unit on left is combination pulverizer and flash dryer with capacity of 500 to 800 lb per hour.

average subbituminous coal requires 291 net Btu per pound for drying, while average lignite requires 429 Btu per pound. Furthermore, if waste heat at 600°F is used for drying, 42 cu ft of hot gas is required per pound of subbituminous coal, and 61 cu ft is required per pound of lignite. On the other hand, if the coals are dried with hot gases at 2000°F, only 7.5 cu ft per pound is required for subbituminous coal and 11 cu ft for lignite.

Thus, the limiting conditions for drying low-rank coals are established both as to time and hot gas required. The object of the experimental work was to find out how close these limits could be approached and to work out technical details of handling materials.

Experimental Work on Flash-drying Fine Coal in Suspension

DESCRIPTION OF PILOT PLANT

By referring to Fig 1, it is observed that ¹⁄₁₆ in. by 0 coal particles can be heated in 1.5 to 2.0 sec. The data of Table 1 show that 42 to 61 cu ft of gases at 600°F is required to furnish enough heat for evaporation of 95 pct of the moisture from subbituminous coal and lignite. These limits on time and gas volume indicate that the coal particles must be suspended in and carried by

the hot gas since the amount of gas required is far beyond the gas volume that can pass through a fixed bed. In order to attain these conditions, the pilot plant and equipment illustrated in Fig 2 and 3 were assembled.

The illustration of the pilot plant is self-explanatory. It consists of a looped vertical column 6 in. in diameter and 19 ft in length. A gas combustion chamber supplies high-temperature products of combustion with minimum excess air, and a fan circulates this gas through the column at the desired rate. By regulation of the amount of gas burned and the rate of circulation, any reasonable variation of hot-gas temperature in the column can be attained, and the rate of circulation in the column is under control. Fine coal dust to be dried is introduced at the base of the column and is carried along with the gas to the separators. Excess gas is discarded after the dust separators, and the temperature of the circulating gases is built up with hot gas from the combustion chamber. The process is continuous, but the dried coal is removed intermittently.

Coal is fed to the drier through the pneumatic feeding system illustrated in Fig 3. In this system the coal is prepared to $\frac{1}{16}$ in. by 0 by crushing through a hammer mill and falls into the dust receiver. The coal is moved from this receiver through a flexible rubber hose by suction. The dust is removed by the "Aerotec" cyclone separator above the charging hopper and the air passes on through a clean-up separator to the exhaust pump. The storage vessel holds 1800 lb of dust, which is sufficient for several hours' operation without recharging. Coal can be charged under pressure continuously by introducing the coal through the double valve arrangement on the charging hopper. The fine dust is metered out of the hopper through a screw conveyor, and the rate of feed is regulated by the variable speed drive. The coal dust is then picked up by a jet of inert gas and transported through a flexible tube to the drying unit. Tubes of different diameter are employed for various charging rates. For example, a $\frac{1}{2}$ in. id tube will handle coal rates up to 200 lb per hour, and a 1 in. tube will handle coal efficiently at rates of 300 to 600 lb per hour. Approximately 1 cu ft of gas is required to transport 1 lb of coal dust in this system, and with the sacrifice of some pressure drop in the line, coal can be moved any desired distance.

Fig 4 shows two flash-drying pilot plants. The structure on the right houses the drier for $\frac{1}{16}$ in. by 0 coal and the larger unit on the left is a combination pulverizer and flash drier. Fig 5 shows the receivers and the circulating gas fan in the lower section of the dust drier.

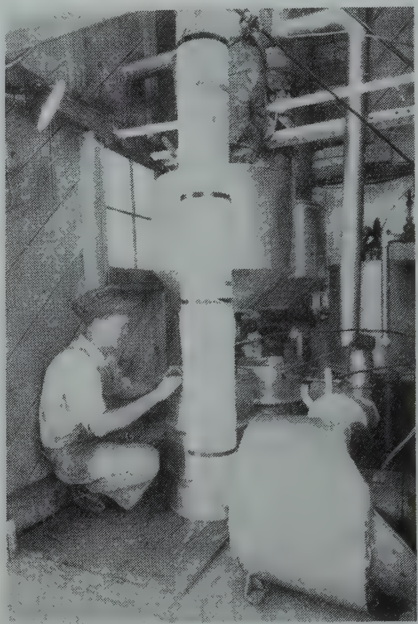

FIG 5—Operating floor of dust flash dryer (right unit in Fig 4) showing dry coal receivers, circulating fan, and combustion chamber.

TESTING PROCEDURE

The drying unit is heated with natural gas several hours before charging coal, and the temperature of the system is regulated to approach desired operating conditions. When thermal balance is established, data are collected to determine normal radiation from the system. The coal is then introduced and the plant operated for a few hours to reach thermal equilibrium. A testing or balanced period then is established, and operating data are recorded. During the balanced period, everything is in equilibrium, and no changes are made. The testing period usually lasts several hours in order to reduce errors to a minimum.

RESULTS OF TESTS AND INTERPRETATION OF DATA ON FLASH DRYING

The operating data and calculations made therefrom are presented in Table 2, which gives the results of 11 tests on flash-drying several coals. As shown from the operating data, coal was handled at rates of 139 to 274 lb per hour, and the percentage of moisture removed D ranged from 54 to 85 pct The rate of processing coal ranged from 690 to 1360 lb per hour per square foot of column area, and the time of contact in the drying column was 1.2 to 1.8 sec, assuming that the coal travels at the same speed as the gas. The heat-balance data indicate that the thermal efficiency is 85 to 92 pct excluding radiation.

The hot gas used for drying varied from 28 to 50 cu ft per pound of coal, depending upon the conditions and percentage of drying. This compares favorably with the minimum hot gas requirements of 42 to 61 cu ft per pound calculated for the removal of 95 pct of the moisture in subbituminous coal and lignite, as previously shown in Table 1. An examination of the data of Table 2 indicates that the process of flash drying is subject to straightforward thermal calculations and the probable performance of a plant on any low-rank coal can be calculated. It is shown that a substantial portion of the moisture is removed in about 1.4 sec, which is in fair agreement with the theoretical time calculated for Fig 1. For a greater degree of drying, it is indicated that the time of contact will have to be extended by lengthening the column or by decreasing the rate of circulation. The relationship of the several factors affecting the flash drying of $\frac{1}{16}$ in. by 0 dusts with low-temperature gases or waste heat can be estimated from the experimental data of Table 2, and it is believed that enough pilot-plant data are presented to warrant design of larger units for particular conditions.

The rate of circulation of gas given in lines 23 and 25 of Table 2 indicate that hot gases should be circulated at a rate of about 10,000 lb per hour per square foot of column area in order to ensure that the $\frac{1}{16}$ in. particles are swept upward with the gas stream. Experimentation on lower rates of circulation indicated that troubles might arise from the larger particles dropping out.

The chemical analyses of the coals before and after drying are given in Table 3, and the screen analysis of the dried product is shown in Table 4. Measurements were made of the bulk density of the dried dusts which indicated the density by ASTM method

to be about 45 lb per cubic foot and the density after tamping to be about 51 lb per cubic foot. During the drying operation, the coal shrinks almost in proportion to the moisture removed which accounts for the relatively high bulk density of the dried product.

Experimental Work on Drying Coal in the Fluidized State

OBJECT AND SCOPE

In a fluidized bed the gas travels at a much greater velocity than the particles in suspension. If the space velocity in a fluidized column at 300°F is 5 ft per second, the system passes 12,300 cu ft of gas per hour per square foot at 60°F. According to Table 1, the hot gas required to dry subbituminous coal is 7.5 cu ft per pound if the gas is supplied at 2000°F. Therefore it is calculated that the capacity of 1 sq ft of the fluidized bed should be 12,300/7.5 = 1640 lb of coal per hour. If the density of the coal in a 6-ft fluidized column is 10 lb per cubic foot, the time of contact is 131 sec, and the velocity of the coal is 0.046 ft per second. Therefore, in the fluidized system considered in this case, the gas travels at 5 ft per second and the coal travels at about 0.05 ft per second, a difference in rate of 1:100.

According to the data of Fig 1, the time required to heat particles of coal ⅛ in. in diameter is about 6 sec, which is less than one twentieth of the time calculated as available in a 6 ft bed. Therefore it is calculated that ⅛ in. by 0 coal should be substantially dried in a 6 ft fluid bed when hot gases are employed at 2000°F. Furthermore, it is indicated that the depth of the fluid bed might be reduced to 1 ft with ample time of contact to remove most of the water from ⅛ in. by 0 subbituminous coal. The object of the experimental work in the pilot plant was to study the system outlined above and to gain experience in handling materials. Various tests are in progress to study the relationship of the several variables.

DESCRIPTION OF PILOT PLANT AND EQUIPMENT

A schematic diagram of the pilot plant for drying coal in the fluidized state is shown in Fig 6, which is drawn

Table 2 . . . Summary Data on Flash-drying Low-rank Coals

Test number	(1)	3	4	5	6	7	8	9	10	11	12	13
Coal name	(2)	Monarch	Monarch	Monarch	Monarch	Monarch	Monarch	Hanna No. 4A	Wyodak	Wyodak	Elkol	Peacock
Materials and moisture data												
Coal charging rate, lb per hr = W_1	(3)	204.0	196.5	203.0	139.0	190.0	191.0	204.0	147.0	189.0	202.0	274.0
Duration of test period, hr[a]	(4)	6.25	2.50	3.50	3.00	7.00	7.00	13.50	9.80	7.40	16.25	14.10
Inert gas used for moving coal, cu ft per lb	(5)	2.0	2.0	0.9	1.3	1.1	1.1	1.3	1.9	1.3	1.3	1.1
Moisture in raw coal, pct (as charged) = M_1	(6)	24.9	25.2	24.6	25.4	16.4	15.2	11.6	30.8	23.6	20.7	12.1
Moisture in dried coal, wtd. avg, pct = M_2	(7)	12.9	13.4	8.8	4.8	2.7	2.8	4.0	6.1	5.6	4.3	3.0
Dry coal recovered, total lb per hr = W_2	(8)	175.8	169.6	165.4	105.5	160.6	162.8	187.0	108.3	152.9	167.3	248.1
Ratio: Coarse to fine coal in receivers.	(9)	2.30	2.49	7.79	6.13	8.8	7.9	5.5	5.0	4.0	5.9	4.4
Improvement ratio by weight loss = R_w	(10)	1.160	1.158	1.227	1.318	1.183	1.173	1.091	1.357	1.236	1.207	1.104
Improvement ratio by moisture determination = R_m	(11)	1.160	1.158	1.210	1.276	1.164	1.146	1.086	1.357	1.236	1.207	1.104
Ultimate improvement ratio = U	(12)	1.332	1.337	1.326	1.340	1.196	1.179	1.131	1.445	1.309	1.261	1.138
Degree of drying, pct = D	(13)	55.3	54.1	70.6	85.2	85.9	83.9	68.3	85.4	78.0	82.9	72.7
Dust loss, percentage of dried coal = L	(14)	0.0	0.0	1.4	3.3	1.6	2.4	0.5	0.0	0.0	0.0	0.0
Heating system data												
Natural gas used, cu ft per hr[b]	(15)	63.3	63.9	79.8	79.2	75.3	78.4	56.7	73.1	73.1	71.4	62.1
Net heat supplied, Btu per hr.	(16)	55,700	56,200	70,224	69,696	66,264	68,992	49,896	64,328	64,328	62,832	54,648
Air used with gas, cu ft per hr	(17)	698	706	830	837	760	789	570	747	748	738	642
Net heat supplied per lb of raw coal, Btu	(18)	273	286	346	501	349	361	245	438	340	311	199
Analysis of circulating gas:												
H_2O, pct (determined by analysis)	(19)	37.0	35.1	47.4	37.6	39.0	40.5	46.0	56.6	57.5	53.7	49.6
CO_2, pct (determined by analysis)	(20)	5.3	5.3	5.4	6.2	6.1	6.1	5.1	4.8	4.5	4.6	4.7
O_2, pct (determined by analysis)	(21)	4.3	4.7	2.1	2.6	2.7	2.5	2.6	1.8	1.5	2.1	2.5
N_2, pct (determined by analysis)	(22)	53.4	54.9	45.1	53.6	52.2	50.9	46.3	36.8	36.5	39.6	43.2
Rate of circulation of heating gas, cu ft per hr	(23)	6,910	6,030	7,240	6,620	6,894	6,798	8,163	7,230	7,125	7,069	7,627
Air leakage from carbon balance, cu ft per hr	(24)	453	456	593	383	419	439	504	729	832	770	640
Mass velocity of circulating gas, lb per hr per sq ft	(25)	2,283	2,008	2,291	2,187	2,266	2,221	2,594	2,194	2,148	2,146	2,832
Contact time, sec (gas circulation)	(26)	1.5	1.8	1.4	1.4	1.3	1.4	1.2	1.3	1.3	1.3	1.2
Temperatures in system, degree F												
Combustion chamber, Point 1	(27)	1,380	1,530	1,775	1,780	1,815	1,790	1,895	1,760	1,740	1,750	1,902
Inlet to drying column, Point 2	(28)	490	440	595	660	650	680	506	659	608	620	573
Midway in "Up" column, Point 3	(29)	290	265	365	440	405	411	363	418	416	408	373
Top of column, Point 4	(30)	237	212	290	355	360	360	333	355	347	355	352
Outlet of column, Point 5	(31)	230	207	285	345	350	350	327	346	335	348	346
Recirculated gas at fan, Point 9	(32)	216	202	267	316	325	325	307	317	312	318	320
Average of points, 2-3-4-5-9	(33)	292	265	360	425	418	426	367	419	404	410	393
Heat balance												
Radiation, Btu per lb raw coal charged[c]	(34)	150	155	150	220	161	160	150	208	161	151	111
Net heat used, Btu per lb raw coal charged[d]	(35)	220	209	277	358	260	250	183	421	325	303	200
Net heat required to dry coal, Btu per lb[e]	(36)	201	192	245	310	224	213	155	364	278	261	174
Drying efficiency, excluding radiation, pct[f]	(37)	91.4	91.9	88.4	86.6	86.2	85.2	84.7	86.5	85.5	86.1	87.0
Overall efficiency of plant, pct[g]	(38)	54.3	52.7	57.1	53.6	53.2	52.0	46.5	57.9	57.2	57.5	55.9
Unaccounted for heat, Btu per lb coal charged[h]	(39)	96	78	81	76	73	49	88	191	145	143	112

[a] Duration of test period, hr. Actual drying operations were carried out for the time indicated. Heat and material data were obtained for a 3-hr balance period (except for test 4, showing a 2½-hr balance period).

[b] Natural gas used, cu ft per hr. All gas and air volumes reported as cu ft per hr, at 60°F and 30 in. Hg, dry. Natural gas for heating has a gross heating value of 985 Btu per cu ft and a net heating value of 880 Btu per cu ft, 60°/30 in., saturated.

[c] Radiation, Btu per lb of raw coal charged. Based on four separate overnight radiation tests at average operating temperatures, a value of 30,500 Btu per hr was established for the overall radiation from the flash-drying unit when operating without coal feed.

[d] Net heat used to dry coal, Btu per lb (calculated). Represents the heat necessary to evaporate water from coal (at 1058 Btu per lb) and raise the temperature of the vapor from 60°F to temperature of the circulated gas, plus sensible heat in dry coal (at 0.25 Btu per lb per °F) from 60°F to temperature of circulated gas plus sensible heat in products of combustion used for pneumatic coal charging between charging temperature and circulated gas temperature plus sensible heat in wet products of combustion rejected to atmosphere between 60°F and the temperature of circulation.

[e] Net heat required to dry coal, Btu per lb. Same as item 35, footnote[d], but less the heat in the wet products of combustion rejected to the atmosphere.

[f] Drying efficiency, excluding radiation, pct. Defined as the net heat required to dry coal, item 36 (see footnote [e]), divided by the net heat to dry plus the sensible heat in the products of combustion and water vapor, item 35, (circulating gas) rejected to the atmosphere times 100.

[g] Overall efficiency of plant, pct. Defined as the net heat required to dry coal, item 36 (footnote [e]), divided by the total net heat supplied to the drying system by the combustion of the natural gas and any coal dust consumed.

[h] Unaccounted for heat, Btu per lb of coal charged. This is the heat liberated by a small quantity of ultrafine coal dust in the bypassed recirculated products of combustion and burned in the combustion chamber. On the average, it represents a negligible loss of material, assuming the heat of combustion of the fine dry coal dust to be in the range of 10,500 to 11,500 Btu per lb, depending on the rank of coal dried. It is calculated from the following values given in Table 2.

$$\text{Unaccounted for heat, Btu per lb coal charged} = \frac{30{,}500 \text{ Btu} - \text{Item (16)}}{\text{Item (3)}} + \text{Item (35)}$$

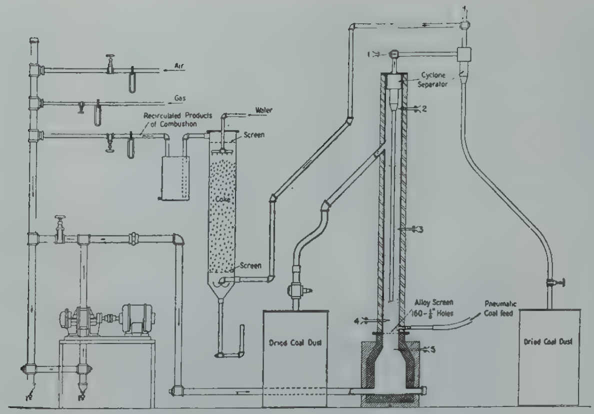

FIG 6—Schematic diagram of fluidized bed drying unit.

to scale. Air, natural gas, and recirculated products of combustion are burned under pressure in the combustion chamber at the base of the column. The combustible mixture is adjusted to generate a temperature of about 2000°F with little excess air. The pump and metering system on the left will deliver any desired quantity of combustible mixture at pressures up to 5 in. Hg. The pilot plant and coal-feeding system is shown in Fig 7.

The products of combustion from the combustion chamber pass through a heat-resisting alloy plate having 160 holes ⅛ in. in size. Coal dust from the metering system previously described and illustrated in Fig 3 enters the column just above the screen and mixes with the hot products of combustion. The bed is fluidized, and the coal moves upwardly to the first offtake and through two dust separators. The dried coal is deposited in the receivers, and the residual gas not entering the recirculating system is discarded.

The drying column is a standard 6-in. steel pipe insulated with mineral wool. Temperatures are taken at 5 points and recorded continuously, and pressure connections are provided at the base and top of the column.

TESTING PROCEDURE

The unit is started by igniting a combustible mixture of air and gas without recirculating products of combustion. When the temperature at point 5 reaches 1700°F, gas recirculation is started and adjustments are made to bring the combustion-chamber temperature to about 1900°F. Coal is then introduced at the desired rate, and adjustments are made to bring the fluidized bed to the selected tempera-

Table 3 . . . Analyses of Coals Before and After Flash-drying

Mine Name[1]	Test No.	Coal Rank	Condition[2]	Analysis, Per Cent									Btu per Lb Coal
				Proximate				Ultimate					
				Moisture	Volatile Matter	Fixed Carbon	Ash	Hydrogen	Carbon	Nitrogen	Oxygen	Sulphur	
Monarch	3	Sub. B	a	24.9	31.8	39.2	4.1	6.6	53.1	1.2	34.6	0.4	9,130
			b	12.9	36.9	45.4	4.8	5.8	61.6	1.3	26.0	0.5	10,588
Monarch	4	Sub. B	a	25.2	31.7	39.0	4.1	6.6	52.9	1.2	34.8	0.4	9,093
			b	13.4	36.7	45.1	4.8	5.9	61.3	1.3	26.3	0.4	10,528
Monarch	5	Sub. B	a	24.6	31.9	39.3	4.2	6.5	53.3	1.2	34.4	0.4	9,166
			b	8.8	38.7	47.5	5.0	5.6	64.5	1.4	23.0	0.5	11,087
Monarch	6	Sub. B	a	25.4	31.6	38.9	4.1	6.6	52.8	1.1	35.0	0.4	9,069
			b	4.8	40.3	49.6	5.3	5.3	67.3	1.5	20.1	0.5	11,573
Monarch	7	Sub. B	a	16.4	35.4	43.6	4.6	6.1	59.1	1.3	28.5	0.4	10,163
			b	2.7	41.2	50.7	5.4	5.2	68.8	1.5	18.6	0.5	11,828
Monarch	8	Sub. B	a	15.2	35.9	44.2	4.7	6.0	60.0	1.3	27.6	0.4	10,309
			b	2.8	41.2	50.6	5.4	5.2	68.7	1.5	18.7	0.5	11,816
Hanna No. 4	9	H. V. Bit. C	a	11.6	40.1	39.8	8.5	5.7	60.4	1.1	23.8	0.5	10,510
			b	4.0	43.5	43.3	9.2	5.2	65.6	1.2	18.3	0.5	11,413
Wyodak	10	Sub. C	a	30.8	30.1	33.2	5.9	6.7	47.1	0.7	39.0	0.6	7,965
			b	6.1	40.9	44.9	8.1	5.1	63.8	1.0	21.2	0.8	10,808
Wyodak	11	Sub. C	a	23.6	33.2	36.6	6.6	6.2	51.9	0.8	33.8	0.7	8,794
			b	5.6	41.1	45.2	8.1	5.1	64.2	1.0	20.8	0.8	10,866
Elkol	12	Sub. B	a	20.7	33.6	43.0	2.7	6.5	58.5	1.0	30.7	0.6	10,248
			b	4.3	40.6	51.9	3.2	5.6	70.6	1.2	18.7	0.7	12,367
Peacock	13	H. V. Bit. C	a	12.1	38.0	45.9	4.0	5.9	66.5	1.4	21.2	1.0	11,675
			b	3.0	42.0	50.6	4.4	5.4	73.4	1.6	14.1	1.1	12,884

[1] Monarch No. 45 mine, Sheridan-Wyoming Coal Co., Monarch bed, Monarch, Sheridan County, Wyo.
Hanna No. 4 mine, Union Pacific Coal Co., No. 2 bed, Hanna, Carbon County, Wyo.
Wyodak mine, Wyodak Coal & Mfg. Co., Smith-Rawlins bed, Gillette, Campbell County, Wyo.
Elkol mine, Kemmerer Coal Co., Adaville bed, Elkol, Lincoln County, Wyo.
Peacock mine, Colony Coal Co., No. 7 bed, Rock Springs, Sweetwater County, Wyo.
[2] Condition *a*, as charged to drier; condition *b*, after flash drying.

ture. Any temperature above 200°F can be maintained in the bed by regulating the amount of heat released or the rate of coal feed. When the system reaches equilibrium and all materials entering are adjusted to constant rates, the testing period is started.

At the start of the test, the dried coal dust receivers are emptied, and the total weight of the coal-charging system is recorded. The unit is then operated several hours under constant conditions, and measurements are made of the rate of flow of all materials, while complete data on temperatures and pressures are recorded. The testing period is ended by reversing the starting procedure. When shutting down the unit, the coal in suspension in the column is determined by quickly shutting off the circulating gas and coal. The coal remaining in the column is then removed and weighed. This measurement is not precise but it has indicated average column densities of 10 to 12 lb per cubic foot.

RESULTS OF TESTS AND INTERPRETATION OF DATA ON FLUIDIZED DRYING

The operating data and calculations made therefrom are presented in Table 5 which gives results from three tests for which complete data were taken. The coal-charging rate during tests 3 and 4 was approximately 1100 lb per hour per square foot which is somewhat under the estimated maximum capacity of 1640 lb per hour. Subsequent tests made on the pilot plant but not reported in this paper have reached capacities of 1500 lb per hour per square foot of column section.

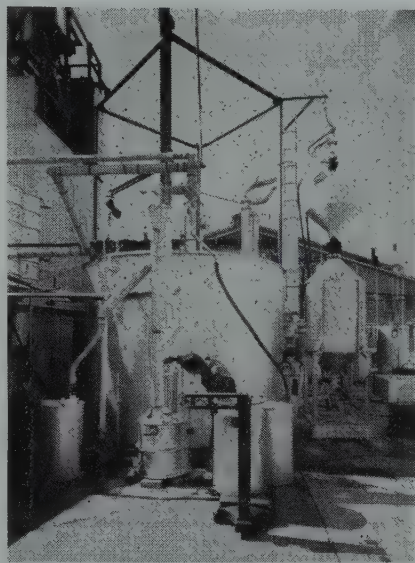

FIG 7—Pilot plant for drying coal in fluidized state. Unit in foreground is 6-in. diam column with drying capacity of 300 lb per hour. Unit on right is coal dust charging system which meters fuel under pressure to pilot plants. The coal dust is transferred pneumatically through flexible conduits.

Table 4 . . . Screen Analyses of Flash-dried Coals[1]

Run No.	4		5		6		7		8	
Sieve, Inch	Retained, Pct	Cum. Retained, Pct	Retained, Pct	Cum. Retained, Pct	Retained, Pct	Cum. Retained, Pct	Retained, Pct	Cum. Retained, Pct	Retained, Pct	Cum. Retained, Pct
(16) 0.0469	0.3	0.3	0.2	0.2	0.1	0.1	0.2	0.2	0.1	0.1
(30) 0.0232	39.5	39.8	38.8	39.0	39.5	39.6	41.1	41.3	34.5	34.6
(50) 0.0117	31.9	71.7	29.4	68.4	29.2	68.8	31.1	72.4	31.1	65.7
(100) 0.0059	12.0	83.7	11.8	80.2	12.9	81.7	13.5	85.9	16.1	81.8
(200) 0.0029	6.4	90.1	9.0	89.2	8.5	90.2	7.4	93.3	9.7	91.5
Pan	9.9	100.0	10.8	100.0	9.8	100.0	6.7	100.0	8.5	100.0
Total	100.0		100.0		100.0		100.0		100.0	

Run No.	9		10		11		12		13	
Sieve, Inch	Retained, Pct	Cum. Retained, Pct	Retained, Pct	Cum. Retained, Pct	Retained, Pct	Cum. Retained, Pct	Retained, Pct	Cum. Retained, Pct	Retained, Pct	Cum. Retained, Pct
(16) 0.0469	0.1	0.1	0.5	0.5	0.8	0.8	0.4	0.4	0.5	0.5
(30) 0.0232	20.1	20.2	34.7	35.2	38.1	38.9	32.8	33.2	32.7	33.2
(50) 0.0117	27.3	47.5	27.3	62.5	22.8	61.7	30.2	63.4	26.8	60.0
(100) 0.0059	19.8	67.3	14.2	76.7	11.0	72.7	16.3	79.7	15.5	75.5
(200) 0.0029	15.9	83.2	11.1	87.8	8.2	80.9	10.8	90.5	9.5	85.0
Pan	16.8	100.0	12.2	100.0	19.1	100.0	9.5	100.0	15.0	100.0
Total	100.0		100.0		100.0		100.0		100.0	

[1] Raw coals crushed to 1/16 in. by 0 in hammermill.
Screen analysis made on composite sample of fine and coarse product from drier.

In test 2, virtually all the water was extracted from the coal since the degree of drying is shown to be 99.1 pct. The dried product from tests 3 and 4 shows that about 94 pct of the moisture was removed, and the total moisture in the dried coal is about 2 pct.

The mean space velocity attained in these tests ranged from 7.5 to 8.8 ft per second, which is somewhat higher than the space velocity of stable fluidized beds. At these high rates, some slugging occurs, and the pressure drop through the column varies from 10 to 20 in. of water but the average pressure drop was approximately 13 in. of water. The mass velocity of the hot gases through the column was approximately 1000 lb per hour per square foot, which is about half the rate employed when drying 1/16 in. by 0 dusts in the flash drier, reported in Table 2.

The heat-balance data are consistent with the theoretical calculations, which indicates that the probable performance of fluidized drying units operating on any coal may be estimated quite closely. The thermal efficiency excluding radiation reaches 86.5 pct and the overall efficiency, including radiation, is over 75 pct. These efficiencies are to be expected in a small unit, but in a large plant the overall efficiency should be 85 to 90 pct.

The physical and chemical properties of the coal dusts before and after drying are shown in Table 6. It is significant that the bulk density of the dried coal is the same as that of the original coal which indicates that the coal particles shrink about in proportion to the loss of weight. This phenomenon has been observed in other experiments on drying low-rank coals. Since the bulk density of the dried product is the same as that of the original coal, the condensation of potential heat, on a volume basis, is in direct proportion to the weight loss. A carload of dried lignite dust will therefore contain about 45 pct more potential heat than a carload of lignite the same size, while average subbituminous coal will improve about 25 pct in potential heat content per cubic foot. This is significant in reducing freight costs on a potential heat basis.

Acknowledgments

This investigation was carried out under the direction of R. L. Brown, Chief, Coal Branch, Bureau of Mines, and in cooperation with H. G. Fisk and C. C. Boley of the University of

Table 5 . . . Summary Data on Fluidized Drying of Subbituminous Coal

Test Number	(1)	2	3	4
Size of Coal	(2)	⅛ in. by 0	⅛ in. by 0	1/16 in. by 0
Materials and moisture data				
Coal charging rate, lb per hr = W_1	(3)	157.7	219.0	227.0
Coal charging rate, lb per hr per sq t	(4)	785	1,090	1,129
Inert gas used for moving coal, cu ft per lb	(5)	1.6	1.2	1.1
Moisture in raw coal, pct (as charged)	(6)	25.0	25.0	21.6
Moisture in dried coal, pct	(7)	0.3	2.0	1.8
Dry coal recovered, total lb per hr = W_2	(8)	116.6	166.5	180.3
Improvement ratio by weight loss = R_w	(9)	1.352	1.315	1.259
Improvement ratio by moisture determination = R_m	(10)	1.329	1.307	1.253
Ultimate improvement ratio = U	(11)	1.333	1.333	1.276
Degree of drying, pct = D	(12)	99.1	94.0	93.4
Dust loss, pct = L	(13)	1.74	0.64	0.49
Heating system data				
Natural gas used, cu ft per hr	(14)	110.8	110.8	109.8
Net heat supplied, Btu per hr	(15)	97,500	97,500	96,600
Air used with gas, cu ft per hr	(16)	1,099	1,099	1,094
Net heat supplied per lb of raw coal	(17)	618	445	426
Hot gas used to dry coal, cu ft per lb	(18)	13.1	9.4	8.9
Products of combustion recirculated, cu ft per hr	(19)	840	840	802
Analysis of gases leaving column:				
H_2O, pct (calculated)	(20)	36.2	41.5	39.7
CO_2, pct	(21)	7.2	6.6	6.7
O_2, pct	(22)	0.8	0.8	0.9
N_2, pct	(23)	55.8	51.1	52.7
Mass velocity in column, lb per hr per sq ft	(24)	972	1,033	988
Mean space velocity in column, ft per sec	(25)	8.8	7.8	7.5
Contact time of coal, sec	(26)	407	289	274
Temperatures in system, °F				
Combustion chamber, point 5	(27)	1,880	1,940	1,910
5 in above screen, point 4	(28)	560	365	380
3 ft, 2 in. above screen, point 3	(29)	540	350	370
6 ft, 11 in. above screen, point 2	(30)	540	350	370
Gas outlet, point 1	(31)	520	335	360
Average bed temperature, points 2, 3 and 4	(32)	545	355	375
Heat balance				
Net heat used, Btu per lb raw coal charged[a]	(33)	543	393	363
Net heat required to dry coal, Btu per lb[b]	(34)	419	340	309
Drying efficiency, excluding radiation, pct[a]	(35)	77.2	86.5	85.1
Overall efficiency, pct[c]	(36)	67.8	76.4	72.5

[a] Total net heat, excluding radiation.
[b] Includes heat used for heating coal carrier gas, excluding radiation.
[c] Includes radiation.

Table 6 . . . Chemical Analyses and Physical Properties of Coal Dried in the Fluidized Bed Drier

Test Number	3		4	
Condition[a]	(1)	(2)	(1)	(2)
Proximate analysis, pct				
Moisture	25.0	2.0	21.6	1.8
Volatile matter	31.3	40.9	32.7	41.0
Fixed carbon	38.3	50.0	40.1	50.2
Ash	5.4	7.1	5.6	7.0
Ultimate analysis, pct				
Hydrogen	6.2	4.7	6.0	4.7
Carbon	51.2	66.9	53.6	67.1
Nitrogen	0.7	0.9	0.7	0.9
Oxygen	36.0	19.7	33.6	19.7
Sulphur	0.5	0.7	0.5	0.6
Btu per lb	8,810	11,510	9,210	11,530
Physical properties. Screen analysis, cumulative pct retained				
On No. 16	15.4	13.3	0.0	0.0
30	46.6	45.3	18.2	17.6
50	71.2	69.8	54.0	53.6
100	85.4	84.0	76.9	75.7
200	94.1	93.2	91.4	89.7
Pan	100.0	100.0	100.0	100.0
Average size, in.	0.0277	0.0265	0.0154	0.0152
Bulk density, ASTM, lb per cu ft	46	46	46	46
Bulk density, impact, lb per cu ft	52	52	52	52

[a] Condition (1) As charged to fluidized drier.
(2) After drying.

Wyoming Natural Resources Research Institute, who conducted experimental work on briquetting the dried dusts.

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Coal Washing in Colorado and New Mexico

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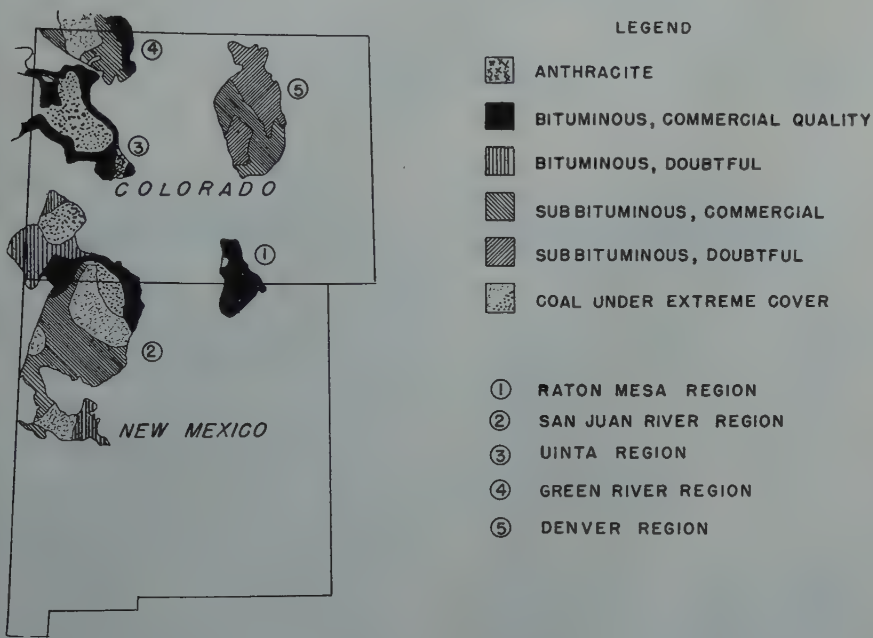

FIG 1—Coal beds of Colorado and New Mexico.

Introduction

In preparing a paper on coal washing in Colorado and New Mexico, it is difficult to refrain from entering into a discussion of the historical aspects of this subject, for the story of coal washing in these states goes back well into the last century. This phase of the subject could well make a complete paper in itself, and of necessity this present discussion must be limited to current operations. Therefore, we are taking into consideration only those washing plants which have operated for some period during the years of 1947 and 1948.

Summary

Of the eight million tons of coal mined in the two states in 1947, approximately two million tons were washed in wet washing plants, using concentrating tables or jigs. The one table plant accounted for 70 pct of the output of washed coal, and two large Baum jig plants accounted for an additional 26 pct, leaving only a small residual tonnage to be accounted for by other types of jigs

The necessity for washing coal is occasioned by the desire or necessity of reducing the ash content of the coals—sulphur is not a problem in this area.

All washeries, except the table plant, are at or near the tipple, hence wash only one kind of coal. At the table plant a large number of different coals are washed. Only at the table plant is there any occasion for drying any considerable quantity of fine coal—other plants size and/or dewater on shaking screens.

The results of washing at the smaller plants are judged by the number (or lack) of complaints about the quality of the washed coal. This is probably as good a system as can be devised for small, intermittent operations.

The efficiency of the Baum jig appears to be quite high. This is to be expected as long as the automatic control of the refuse gate operates properly. The efficiency of the concentrating table is not as high as that of the Baum jig, especially when the "difficulty" of washing is taken into consideration. When dealing with coals in which the "bone" is larger than the "coal," the table appears to be at a definite disadvantage, because of its sizing action.

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TP 2548 F. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before June 30, 1949.

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Where Is the Coal in These States?

It was estimated by Marius R. Campbell[1] that the original tonnage of coal in these two states was as shown in Table 1.

Table 1 . . . Original Tonnage of Coal in Colorado and New Mexico

State	Subbituminous, Tons	Bituminous, Tons
Colorado....	104,175,000,000	213,071,000,000
New Mexico.	172,906,000,000	18,925,000,000

The figure shown for Colorado bituminous coal is larger than that for any other state in the United States, and the figures for subbituminous coal are exceeded only by that for Wyoming. The quantity of commercially minable coal in this area is probably not as large as these figures might be interpreted to indicate; but that is another story.

A map locating the coal beds of these states (Fig 1) shows them as lying in five general "regions."

1. The Raton Mesa region of Colorado and New Mexico, lying across the border of the two states in about a central location and containing the only heavy-coking coal to be found in these states.

2. The San Juan River region, also lying across the border but near the western edge of the states, which contains weakly coking bituminous coals in some areas in addition to a large quantity of subbituminous coal.

3. The Uinta region, lying across the border of Utah and Colorado, also contains weakly coking bituminous coal in certain areas, as well as noncoking bituminous coal and some anthracite.

4. The Green River region, extending from Wyoming into Colorado, contains noncoking bituminous and subbituminous coals.

5. The Denver region, being wholly within the north-central portion of Colorado, contains only low rank subbituminous coals approaching lignite in appearance and quality.

Within each region are a number of fields and districts. Coal washing is, at present, confined to the Trinidad, Raton, and Yampa fields. Taken together, the Trinidad and Raton fields comprise the Raton Mesa region; the Yampa field is on the southeastern border of the Green River region in Routt County, Colo. Fig 2*A* and 2*B* show generalized sections of the coal measures in these two localities.

[1] References are at the end of the paper.

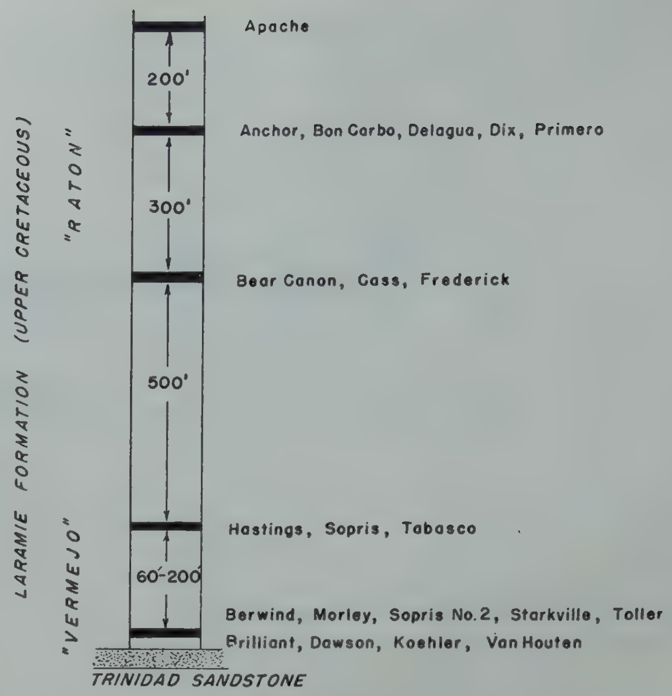

FIG 2*a*—Coal beds of Trinidad-Raton area, southern Colorado-northern New Mexico.

Typical proximate analyses of some of the coals now being washed are given in Table 2.

Table 2 . . . Typical Analyses of Colorado and New Mexico Coals

Field	Moisture-free Proximate Analysis		
	Volatile Matter	Fixed Carbon	Ash
Trinidad field, Colorado...	30.2	51.7	18.1
Trinidad field, Colorado...	31.6	50.8	17.6
Raton field, New Mexico..	35.7	45.6	18.7
Raton field, New Mexico..	35.1	45.2	19.7
Yampa field, Colorado....	41.5	54.5	4.0
Yampa field, Colorado....	41.4	50.8	7.8
Yampa field, Colorado....	40.6	49.1	10.3

Who Is Washing Coal?

The operating coal washeries in the two states are owned by the following companies:

1. The Phelps Dodge Corporation is operating a Link-Belt Simon-Carves 84-in. 2-compartment pneumatic jig at the Stag Canon Branch, Dawson, New Mexico, and is washing Raton field coking coal.

2. The St. Louis, Rocky Mountain and Pacific Company is operating a Jeffrey 84-in. 2-compartment pneumatic jig at the Koehler mine, and is washing Raton field coking coal.

3. The Gordon Fuel Company has a single compartment 36 by 60-in. basket type jig at each of the tipples at the Gordon and Alamo No. 2 mine, near Walsenburg, Colo., at the northern edge of the Trinidad field. This is noncoking coal.

4. While not located in the Raton Mesa region, the Colorado Fuel and Iron Corporation has a washing plant at Pueblo, Colo., consisting of 24 Deister Plat-O tables and is washing coal mainly from mines in the Raton and Trinidad fields.

5. Plants located in the Yampa field (Green River region) of Routt County, Colo., include the Edna Coal Company, operating a 60-in. single compartment Jeffrey diaphragm jig on coal from a strip mine near Oak Creek, Colo.

6. The Moffat Coal Company also operates a 60-in. single compartment Jeffrey diaphragm jig at Routt, Colo.

7. The Keystone Coal Company has a similar jig in operation in the same locality.

8. The Victor-American Fuel Company is operating two plants in the Green River region. At the Pinnacle plant, at Oak Creek, three 30 by 48-in. plunger type jigs of their own design and manufacture are in operation. At the Wadge plant, at Mount Harris, there are two jigs of the same design and size.

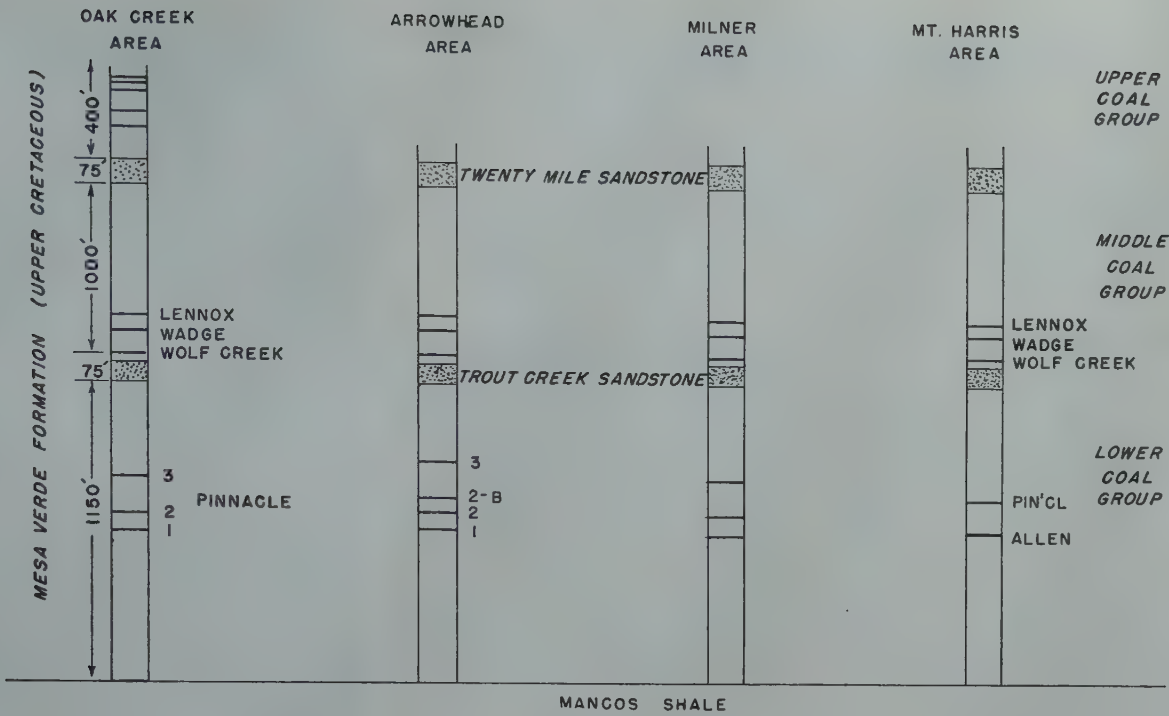

FIG 2b—Generalized sections of coal measures, Routt County, Colorado.

How Much Coal Is Being Washed?

With a total mine output of approximately eight million tons, the washeries in the two states handled approximately two million tons, in the year 1947. In Table 3 certain statistics on coal mined and coal washed are shown.

In each state there is one washery which accounts for a very large percentage of the coal washed: in Colorado, the Pueblo washery accounts for approximately 95 pct of the coal washed, and, in New Mexico, the Dawson washery accounts for approximately 85 pct of the coal washed. Other washeries are either of small capacity and/or are operated only as market requirements demand.

Why Is Coal Being Washed?

Coal is being washed in these states for two primary reasons:

1. To improve the chemical characteristics of coals used for coke production and other industrial purposes.
2. To improve the appearance and to some degree the quality of free burning coals produced for the domestic trade.

Three of the ten plants are operating on high volatile bituminous coking coal, cleaning coal mined in the Raton Mesa region (see Fig 1). In these plants, in which 96 pct of the coal washed in these states was handled, the necessity for cleaning the coal is increased by the considerable increase in mechanized mining, and, at one plant, by the inauguration of full seam mining. These three plants are, as would be expected, in or fairly close to the Raton Mesa region.

Table 3 . . . Mine Output and Coal Washed, Colorado and New Mexico, 1947 Totals

	Tons	Per Cent
Total coal mined:		
Colorado	6,371,132	80.6
New Mexico	1,534,400	19.4
	7,905,532	100.0
Colorado coal to washeries, per cent		15.2
New Mexico coal to washeries, per cent:		
Colorado		31.6
New Mexico		36.3
		67.9
Coal washed in Colorado:		
from Colorado mines	967,333	62.2
from New Mexico mines	484,411	31.1
from Oklahoma-Arkansas mines	103,692	6.7
	1,555,436	100.0
Coal washed in New Mexico:		
from New Mexico mines	556,508	100.0
Total coal washed:		
Colorado	1,555,436	73.7
New Mexico	556,508	26.3
	2,111,944	100.0

Of the other seven plants, washing only the nut size (3 by 1¼ in.) of coal produced mainly for the domestic trade, two are in the Trinidad field, Huerfano County, and five are in the Yampa field, Routt County, Colo.

All plants are primarily interested in ash reduction, whether actually to improve the chemical quality of the coal or merely to improve its appearance. None of them are particularly concerned with sulphur reduction, the sulphur content of coal produced in these states being so low (seldom exceeding 0.6 pct) that it is necessary to give this impurity but little attention. Because of this fact, this paper will deal entirely with the ash reduction practice of these plants.

What Methods of Preparation Are Being Used?

ADVANCE PREPARATION OF COAL

Nine of the washing plants are located at or near the tipple, and handle coal from the local source only.

The largest of this group of washing plants (Dawson) washes the entire mine output (full seam) with the exception of hand-picked plus 6-in. rock. Run-of-mine coal is dumped by a rotary mine car dumper on 6-in. grizzlies—the oversize is picked and the large coal is crushed to pass 6-in. and returned to the main stream. Just ahead of the jig the minus ⅜-in. is screened out on a

Table 4 . . . Preliminary Preparation of Coal Sent to Washing Plants in Colorado and New Mexico, 1947

	Plants	Tons	Per Cent
Coal crushed (or screened) at mine to meet slack classification. Further crushed at plant to minus ½ in.	1	1,472,953	70
R-O-M screened at 6-in., oversize picked and crushed, minus ⅜-in. removed	1	485,000	23
R-O-M screened to 3 by 1¼-in.	7	82,483	4
R-O-M screened to 3 by ⅜-in.	1	71,508	3

vibrating screen and bypassed, leaving a 6 by ⅜-in. jig feed.

The other mines practice selective mining to a large extent, and much of the rock is gobbed in the mine. In some cases it is necessary to pick bone. Run-of-mine coal is passed over shaker screens and four or five sizes are usually produced. Of these sizes the *nut* or *nut-and-pea* are washed. Two of the tipples are so arranged and equipped that large sizes may be crushed, permitting operation even though there are no orders for lump coal.

The shaking screens are of various types and sizes. Some are suspended from steel rods, others by wooden "slats," and some are supported by wheels on tracks. All screening surfaces are punched plate. Ordinarily, provision is made to combine several sizes, as desired, either by blanking off the screen or by recombining the screened coal. The crushers, where used at the tipples, are of the single or double toothed-roll type.

One washery is located so that it may be used as a central washing plant for several mines, if that should prove desirable. At present it is equipped with a vibrating screen ahead of the feed bin and bypasses the fines. When the washery was built, sufficient room was left for the installation of drying equipment so that, if necessary, they could wash *nut-slack* or any other "through" size ordered.

The Pueblo plant of the Colorado Fuel and Iron Corp. is a central cleaning plant for a number of coals. Since coals from different sources vary not only in physical and chemical characteristics but are different in coking properties, one of the greatest problems at this plant is maintaining a uniform mixture, for the delivery of these coals is anything but uniform. Coals are dumped and crushed either as individuals or in "pairs" as members of a "family" of coals, then stored temporarily in mixing bins, from which they are drawn as needed. With a limited number of bins, the resulting mixture is not necessarily uniform from the beginning to the end of a shift, nor will it average out from shift to shift.

All coal used at this plant is screened or crushed to meet the *slack* classification of the railroads (minus 1½-in. or minus 2-in.) before loading; but at the plant is further crushed to approximately 85 pct through ½-in. for the purpose of liberating the maximum amount of free impurity.

Table 4 summarizes the preliminary preparation of coal to the washing plants for the year of 1947.

COAL WASHING EQUIPMENT

As previously noted, only two general types of washing equipment were used in this area in 1947: the concentrating table, and the jig.

The table plant has been in operation since 1922 and is beginning to show signs of old age in several places. The Baum jigs are fairly new and apparently still in good shape. They were installed in 1945 and 1946. One of the diaphragm jigs was installed in 1947, and the others are relatively new. The plunger jigs are less than five years old. They are steel duplicates of the worn-out wood-tank jigs they replace. The basket jigs are of uncertain age; but are obviously much older than is indicated by the fact that they were put into service in their present location in 1937.

Arrangement of the washing plants ranges from excellent to terrible. Those plants in which the washery is in a separate building generally are well arranged, although the lighting could be improved upon in some cases, and a little more "elbow room" would not be unwelcome. Where the washing unit is simply an appendage to the tipple, there is usually room for considerable improvement in many respects. Generally speaking, the best work is done in the best surroundings. If one is to judge solely by the appearance of the product, there must be good illumination to turn out a good product.

Table 5 gives the tonnages and percentages of coal washed in the various types of equipment in Colorado and New Mexico for the year 1947. Where the washery operators were unable to furnish information, we have relied on the report of the State Coal Mine Inspector.

Table 5 . . . Coal Washed in Various Types of Equipment, Colorado and New Mexico, 1947

Type of Washer	Number of Plants	Tons Washed	Percentage of Total
Concentrating tables, wet	1	1,472,953	69.74
Baum jig	2	556,508	26.35
Diaphragm jig	3	31,291	1.48
Plunger jig	2	33,102	1.57
Basket jig	2	18,090	0.86
Total	10	2,111,944	100.00

HANDLING SUBSEQUENT TO WASHING

The drying of coal at those plants having a sized feed is a comparatively simple matter. Vibrating or shaking screens, provided with a wash spray of clear water, will dewater sufficiently to permit coal to be loaded directly into railroad cars for shipment to the consumer. Such dewatering screens also remove any degradation products, the washed fines being, in some cases, sold as stoker pea. In other cases the washery fines are loaded out with dry slack (which has bypassed the washery).

One plant is so arranged that the washing unit is mounted over loading bins and all sizes of dewatered washed coal are run directly to these bins. Because of this arrangement this plant is able to load out washed coal to either cars or trucks with equal facility.

The dewatering of coal at the table plant, where the coal is more finely crushed, is more difficult. A large portion of the minus ⅛-in. coal is screened out and dewatered on vibrating filters. The coarse coal is passed through a battery of Carpenter centrifugals and then all dried coal is combined and taken to storage bins for charging into the coke ovens. The coarse coal from the Carpenters will average about 7 pct moisture and the fine coal from the vibrators will run about 22 pct, giving a final ⅜ by 0 dewatered product containing about 11.5 pct moisture.

This seemingly high percentage of

water in the coal to the ovens is actually beneficial in several respects. It results in an increased bulk density and higher net coal charge per oven than any other lower moisture content above 3 pct, retards carbon formation on the roofs and walls of the ovens, and increases the yield of ammonia from the coal. The action of moisture content of coal upon its bulk density is illustrated in Fig 3.

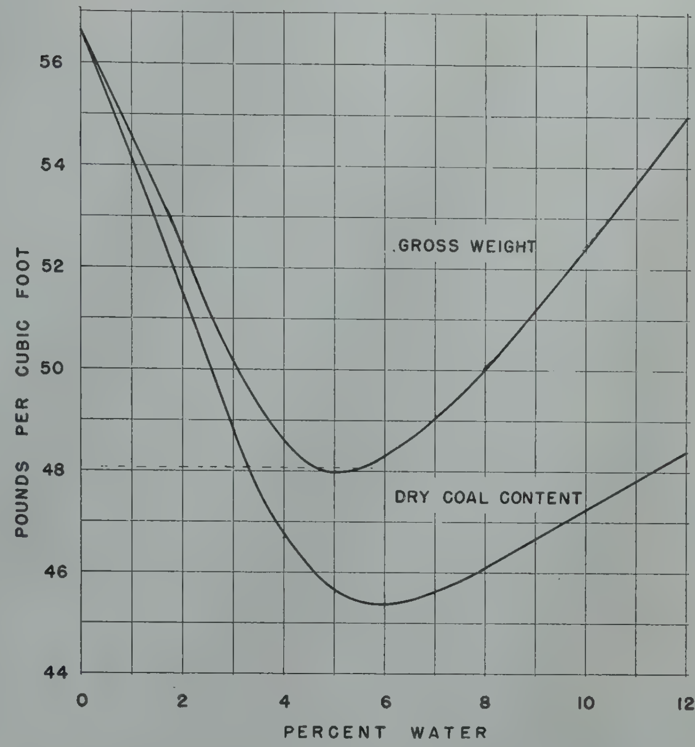

FIG 3—Effect of moisture content on coal weight per cubic foot.

What Results Are Being Secured?

Since there are a number of factors influencing the general conduct of coal washing operations, it would appear advisable to consider certain of these at this point.

One of these factors is largely psychological and may be introduced with the question, "Is the washing of this coal really necessary?" The more convincing the arguments in favor of washing a particular coal the more likely we are to find good results, regardless of the actual equipment used—although we will usually also find better equipment in use than where there is room for doubt as to the necessity of washing. Another intangible factor is tradition or reputation—if a company has managed to build up a good reputation for producing as good a coal as can be found in its territory, it is quite likely that considerable effort will be put forth to maintain this reputation.

On the other hand, we have the more tangible factors of washability characteristics and type of equipment used. Coals of the easy-to-wash type may be cleaned quite efficiently with rather simple equipment and very little technical supervision. The use of elaborate equipment and "quality control" facilities in some of the installations here considered would probably be an economic waste. Other coals probably could well be treated more scientifically than they are with gratifying returns on the investment.

Each washery has its own background and current problems, and it is not our intention to attempt to justify or condemn the various practices we found. Our role is that of observer, not consulting expert.

Samples were obtained from each operating plant; these were all tested and analyzed at one central laboratory.[2] The observations which follow are based on the data so secured and the practice at the various plants will be discussed by geographical groups rather than by individual plants.

HUERFANO COUNTY, COLORADO

The coal washed in this county represents the smallest amount of any of the geographical areas in the two states. Two plants in this county are operated intermittently as market requirements dictate. Samples obtained from the one currently operating plant indicate results on products as given in Table 6. Information as to washery yield and as to character of the feed coal is lacking.

Table 6 . . . Float-sink Data, 3 by 1¼ In. Nut Coal, Huerfano County, Washery A

Specific Gravity	Washed Nut		Reject	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
1.30	13.8	6.9	3.3	5.6
1.38	62.7	9.6	6.2	8.9
1.45	15.4	17.8	4.2	17.5
1.50	4.1	25.0	2.7	24.3
1.60	2.1	32.7	4.4	33.0
1.70	1.4	43.0	3.4	43.6
Sink	0.5	59.6	75.8	78.1
Cumulative ash		12.3		64.3

As in the case of several other washeries, the operation is irregular and not subject to laboratory control. Washing has removed practically all high gravity components from the washed coal and makes it an acceptable commercial product.

ROUTT COUNTY, COLORADO

All of the washeries in this county operate intermittently as the demand for washed nut (3 by 1¼ in.) coal for domestic use may require. Samples of the feed coal and of the products from four of the five plants were obtained and tested in the central laboratory, the fifth plant was idle for a considerable period during the current year. Results on washed coal samples from these plants are as shown in Table 7.

Although each of these plants operates under somewhat different conditions, on feed coal of different washability, and with products of differing quality, one plant only has been selected for discussion and to represent the practice of this county. Data on the feed coal, washed coal and reject from Washery *E* are shown in Table 8.

As previously noted, certain plants in Colorado and New Mexico wash coal mainly to improve the appearance of the product. This is particularly true of Routt County. In the case of Washery *E* the ash content has been reduced from 10.7 to 8.0 pct. This was done principally by the removal of high ash impurity, the sink in 1.6 in the feed coal was 6.6 pct and this was reduced to 1.8 pct in the washed coal. In round numbers, three-fourths of the rock was

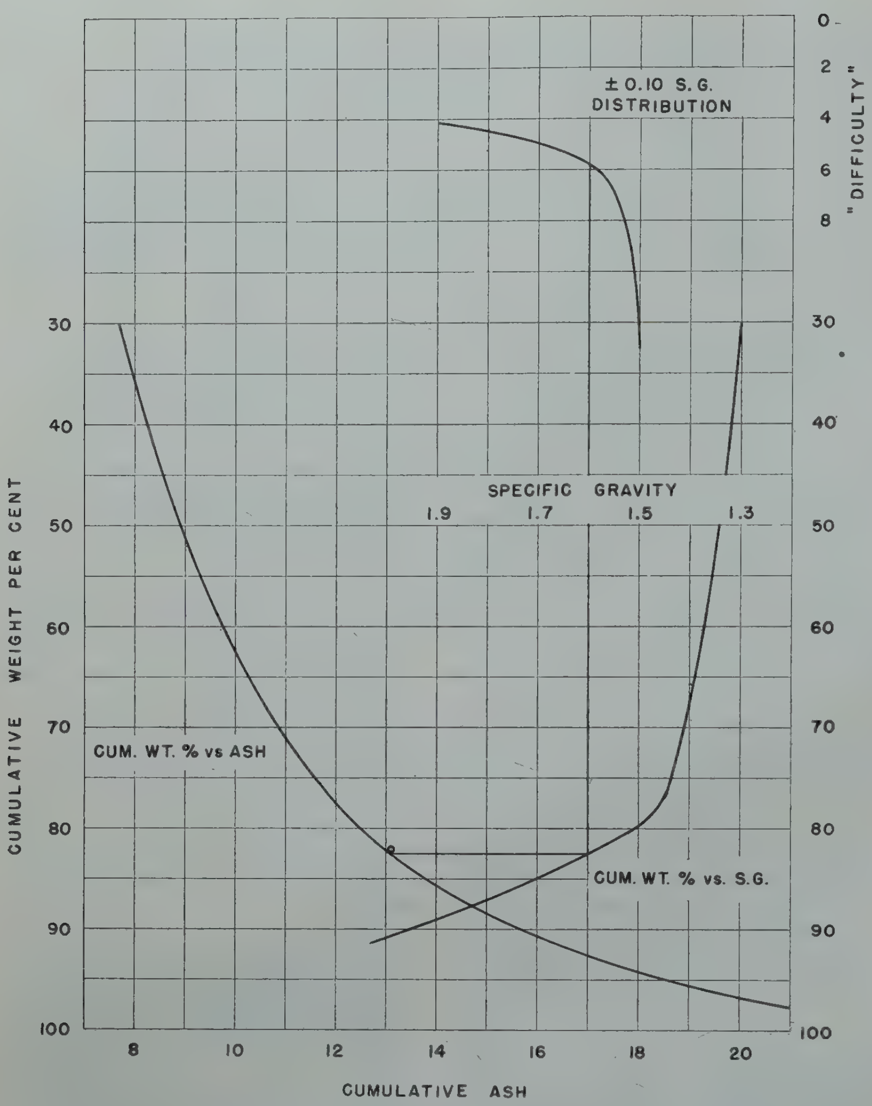

FIG 4—Washability curves, washery *H*, Colfax County, New Mexico.

Table 7 . . . Float-sink Data, Washed Coals, Routt County

Specific Gravity	Washery *C*		Washery *D*		Washery *E*		Washery *F*	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
1.30	79.2	3.5	67.6	3.0	73.2	4.2	84.2	2.6
1.38	15.7	8.9	18.2	10.1	15.8	9.1	5.0	9.5
1.45	1.6	19.3	10.0	14.6	4.0	19.8	2.3	13.8
1.50	2.1	27.3	1.1	27.6	2.9	26.7	0.4	27.3
1.60	0.6	37.0	0.8	34.2	2.3	34.5	0.3	34.6
1.70	0.6	43.5	0.9	45.1	0.4	39.4	0.5	36.4
Sink	0.2	57.2	1.4	69.3	1.4	69.0	7.3	70.4

Table 8 . . . Float-sink Data, Washery E, Routt County

Specific Gravity	Raw Coal		Washed Coal		Reject	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
1.30	72.6	4.2	73.2	4.2	2.3	4.5
1.38	12.5	9.0	15.8	9.1	1.4	9.0
1.45	4.1	19.7	4.0	19.8	0.8	19.5
1.50	3.0	26.7	2.9	26.7	0.9	26.7
1.60	1.2	35.5	2.3	34.5	2.7	35.8
1.70	1.0	41.3	0.4	39.4	2.8	45.5
Sink	5.6	72.2	1.4	69.0	89.1	75.0
Cumulative ash		10.7		8.0		69.7

removed. Apparently this is sufficient to minimize complaints and maintain their position in the market.

With none of these plants washing as much as 20,000 tons of coal per year, it does not appear that it would pay to install laboratory facilities and inaugurate a system of quality control at an individual washery. While exact figures are not available, yields at all washeries appear to be above 90 pct. The coal that might be saved by increased efficiency because of scientific control as compared to visual inspection probably would not be worth enough to warrant much of an increase in expenditure along this line.

COLFAX COUNTY, NEW MEXICO

There are two plants operating in this county, one washing the entire mine output as a matter of routine, the other washing only a portion of the coal on special order. The two situations are not comparable. One mine takes the full seam, and the coal is shot down. The other mine loads rock separately as much as practicable and uses airdox.

At both of these washeries samples were secured which were representative of the different products, as well as estimates of the relative proportions of all components. These individual samples have been combined together as shown in Table 9 as the calculated feed for Washery *H* and in Table 12 as the calculated feed for Washery *J*.

At Washery *H* the 6 by ⅜-in. feed is summarized in Table 9 as to weight per cent and ash content at various specific gravities. The raw coal is screened at 6 in. and the large rock is hand picked, after which the lump coal is crushed to pass 6 in. Just ahead of the jig the minus ⅜ in. is screened out and bypasses the washbox. Note the small amount of float-on-1.3 material, the small amount of middle gravity material and the high ash content of the heavy rock.

The recovery from the Baum jig is 82 pct of washed coal with an ash content of 13.1 pct and 18 pct of refuse with 71.1 pct ash. This indicates an efficiency of 99.4 pct, and represents very good practice. In this discussion "efficiency" is the percentage ratio of washed coal recovery to the recovery possible with a perfect float-sink separation at the same ash content.

Tables 10 and 11 show the float-sink data for the various sizes of washed coal and the reject from the two compartments. These results show the presence of very little heavy gravity material in

any of the washed coal samples and comparatively little good coal in either of the reject samples. It would be difficult to suggest any improvement in this operation.

Table 9 . . . Calculated Feed to Washery H, Colfax County

Specific Gravity	Weight, Per Cent	Ash, Per Cent	Cumulative Weight	Cumulative Ash
1.30	30.1	7.7	30.1	7.7
1.38	30.2	11.7	60.3	9.7
1.45	16.9	20.2	77.2	12.0
1.50	2.6	25.7	79.8	12.4
1.60	2.6	33.3	82.4	13.1
1.70	2.6	45.3	85.0	14.1
Sink	15.0	77.4	100.0	23.6

One of the noteworthy characteristics of this coal is that even after excellent washing it is still as high in ash as many of the eastern raw coals (with careful mining practice). This is true of much of the Raton Mesa coal.

In Fig 4 we show the washability curves for this coal. Note that this coal is being washed at a plus or minus 0.1 sp gr *difficulty* of about 6 pct and that it would be difficult to reduce the ash in the washed coal much further because of the extremely rapid increase in *difficulty*. There is no sharp break in the float-ash curve of the feed coal, although there are such breaks in the curve of cumulative weight vs. specific gravity and the *difficulty* curve.

At Washery *J* in this county the feed is 3 by ⅜ in. and may be one of two types: (1) natural minus 3 in., or (2) modified mine run. Normal operation at this mine sends very little heavy rock to the tipple; but large "bone" is picked before a feed of type 2 is crushed for the washery, hence there is little extremely high gravity material present in the feed coal.

The composite of all products indicates the washery feed is as shown in Table 12. Note the low ash of the material sinking in 1.7, as well as the uniform increase in cumulative float ash.

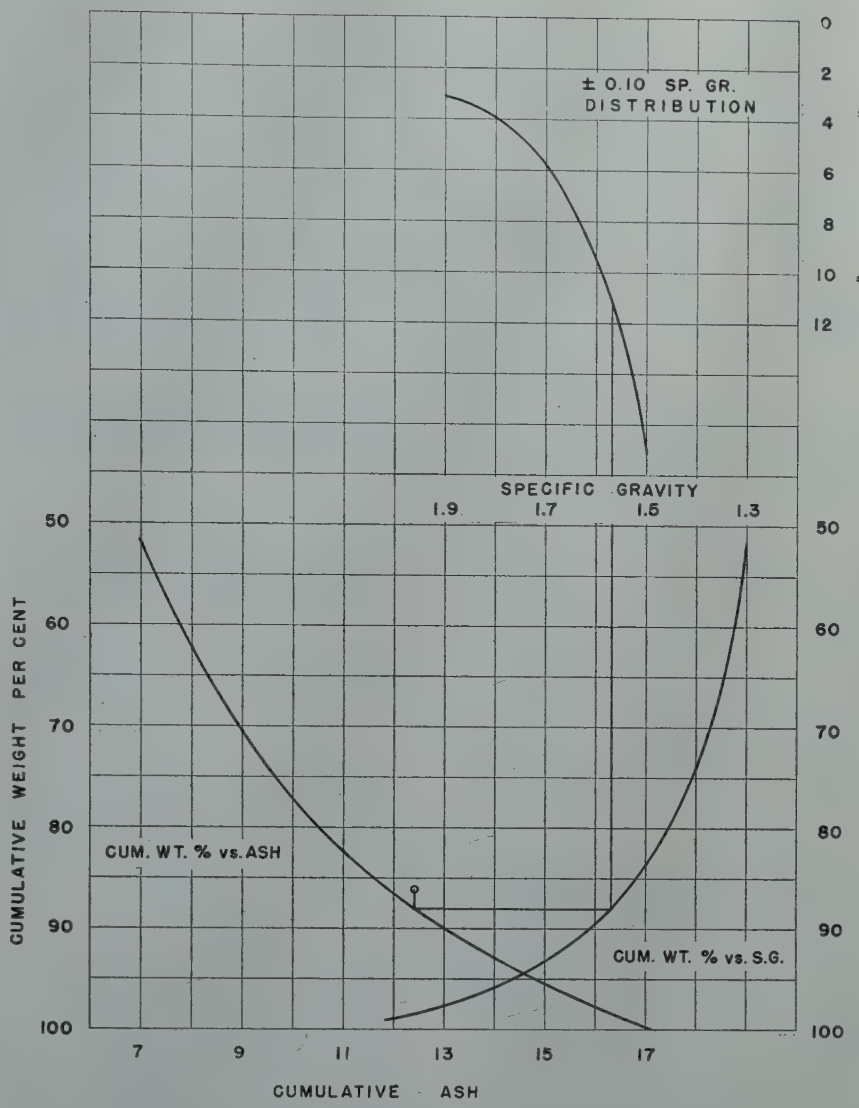

FIG 5—Washability curves, washery *J*, Colfax County, New Mexico.

From this feed a recovery of 86 pct of washed coal of 12.4 pct ash is made. The reject contains only 46.2 pct ash; but the efficiency is 97.7 pct.

The washability curves shown in Fig 5 indicate a much higher *difficulty* than was the case for Washery *H*. In addition to this we find a larger amount of free coal in the minus 1½-in. portion of the reject than was the case for Washery *H*. This condition was not notice-

Table 10 . . . Float-sink Analyses of Various Sizes of Washed Coal from Washery H

Specific Gravity	6 by 3 In.		3 by 1½ In.		1½ by 1 In.		1 by ⅜ In.	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
1.30	15.5	7.2	35.2	8.5	62.3	7.1	48.5	7.3
1.38	58.1	10.1	30.6	12.4	25.2	12.8	31.6	12.7
1.45	22.4	20.1	27.3	20.4	5.3	20.4	12.9	20.7
1.50	0.5	27.4	4.4	24.9	2.8	27.0	3.1	26.5
1.60	3.3	31.5	2.3	31.9	3.7	35.0	2.4	34.0
1.70	0.1	37.3	0.1	38.5	0.5	44.7	1.0	40.6
Sink	0.1	47.2	0.1	65.8	0.2	58.2	0.5	57.1
Cumulative ash		12.7		14.3		11.1		12.6

Table 11 . . . Float-sink Analyses of Reject H

Specific Gravity	First Cell		Second Cell	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
1.30	0.6	8.5	0.1	8.0
1.38	1.3	12.5	0.4	12.5
1.45	1.8	21.0	0.5	23.5
1.50	0.8	28.2	0.3	29.0
1.60	1.1	35.9	3.1	37.4
1.70	3.8	48.0	21.4	45.5
Sink	90.6	81.1	74.2	73.5
Cumulative ash		76.5		65.7

Table 12 . . . Calculated Feed to Washery J, Colfax County

Specific Gravity	Weight, Per Cent	Ash, Per Cent	Cumulative Weight	Cumulative Ash
1.30	51.6	7.0	51.6	7.0
1.38	17.3	14.2	68.9	8.8
1.45	9.8	20.9	78.8	10.3
1.50	4.8	27.3	83.6	11.3
1.60	5.9	34.3	89.5	12.8
1.70	3.5	42.3	93.0	13.9
Sink	7.0	59.7	100.0	17.1

Table 13 . . . Float-sink Analyses of Washed Coal J, Colfax County

Specific Gravity	3 by 1½ In.		1½ by 1 In.		1 by ⅜ In.	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
1.30	44.7	7.7	57.6	7.6	69.6	6.5
1.38	29.3	15.0	20.5	14.2	14.0	13.3
1.45	17.5	21.8	10.8	21.1	7.2	19.5
1.50	5.8	28.2	5.9	27.0	3.6	25.5
1.60	2.0	33.5	4.1	34.0	2.9	33.5
1.70	0.6	39.6	0.6	39.5	1.3	40.3
Sink	0.1	57.1	0.5	50.3	1.4	51.5
Cumulative ash		14.3		13.0		10.9

able in the reject elevator of the jig, and we have been unable to determine the exact source of this low-gravity material. Crushing tests on the large coal and large reject indicated that little could be done in the way of further liberation of either coal or impurity. We believe that this situation indicates that there is a very small percentage of the coal which is attached loosely to high gravity material and that but little rough handling is required to liberate this coal. Drastic crushing certainly is not indicated as the solution to the problem.

In Tables 13 and 14 we show the float-sink analyses of the products from the jig. The washed coal is sized by the dewatering screens of the plant. The reject normally is not sized; but a size separation was made before running the float-sink tests, to determine the difference between large and small reject.

The decrease in ash of the washed coal as the size decreases is normal for this coal. No such trend is evident at Washery *H*. Experiments on crushing the 3 by 1½-in. washed coal (and reject) indicated that even though this portion were crushed to pass ¾ in. its washability characteristics would not be improved noticeably. The large material is inherently more bony than the small, which is probably because of differences in strength of bone and coal, and there is little that can be done about it.

Table 14 . . . Float-sink Analyses of Reject J, Colfax County

Specific Gravity	3 by 1½ In.		Minus 1½ In.	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
1.30			3.4	7.2
1.38	0.4	15.7	2.5	11.6
1.45	0.8	25.5	5.4	20.3
1.50	5.5	29.5	4.7	26.6
1.60	36.4	36.0	14.6	32.1
1.70	21.2	43.8	17.0	42.2
Sink	35.7	56.8	52.4	62.8
Cumulative ash		44.6		47.6

PUEBLO COUNTY, COLORADO

This washing plant is operated in conjunction with the manufacture of metallurgical coke, and since no single source of supply is capable of meeting our requirements, coal from a large number of mines is washed—sometimes as many as 15 in a single month, and as many as 40 in the course of a year.

This situation is not at all favorable. There are constant changes in the character of the feed to the washery, not only from day to day; but from shift to shift and quite probably from hour to hour. The use of yearly averages in such a case is inadequate; but can hardly be avoided in a brief discussion.

As an example of the possibility for variation, individual cars of coal 2 in Table 15 have been known to be as low as 18 pct ash and some have been as high as 40 pct ash. Coal 1, on the other hand, is more uniform, the range being about 5 pct in ash. Other coals probably have a range not exceeding 7 or 8 pct with ordinary care in mine preparation. We could, presumably, obtain a fairly uniform mixture if all coals were received daily in the same proportion as ordered for the week; but this is a situation that has not developed yet. The blending system does eliminate the extreme variations; but, because of limited capacity, cannot furnish a truly uniform feed.

Table 15 shows the 1947 average float-sink data on all major coals (1 to 6) and two groups of coals (7 and 8). Coals 1 to 6 are all high-volatile coking coals. Coals 1 and 2 are from the same mine, 1 being screened slack and 2 crushed lump. Coal 3 is from a company-owned mine in the same field but on a different seam. Coals 4, 5, and 6 are purchased from New Mexico mines in the same region, and from the same seam as coal 3.

Group 7 is the composite of about 25 coals from the low-volatile field of Oklahoma-Arkansas. When available in sufficient quantity, this group is used as 10 pct of the normal mixture. A small amount of special coke is made with 25 pct of these coals in the mixture.

Group 8 is the composite of a number of high-volatile coals which are purchased occasionally to meet our tonnage requirements. The greater part of this group is noncoking coal.

Table 15 is made up from the analy-

Table 15 . . . Float-sink Data of Individual Coals Used at Pueblo, Yearly Averages, 1947

Coal	1		2		3		4		5		6		7		8	
Percentage of Mix	26.11		14.03		13.28		13.53		11.10		8.26		7.04		6.65	
Specific Gravity	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
1.30	76.2	5.8	48.7	6.4	53.2	7.3	53.0	6.5	51.2	6.1	56.3	5.8	66.4	3.6	61.0	6.0
1.38	8.0	13.4	14.2	12.6	19.6	14.5	13.6	14.5	16.5	13.2	13.2	12.6	12.4	8.2	16.2	12.4
1.45	3.3	20.2	6.1	20.0	9.3	21.4	7.3	22.2	6.9	21.4	6.4	20.8	4.2	14.3	5.0	19.8
1.50	1.3	27.2	2.5	27.0	3.6	27.4	4.3	28.5	3.7	28.1	4.2	27.1	1.8	20.4	2.6	25.9
1.60	1.3	34.4	3.1	34.0	3.9	34.3	4.9	35.2	5.2	35.0	5.5	33.8	2.2	26.5	2.7	34.0
1.70	1.0	43.1	2.3	42.8	2.6	42.7	3.4	43.4	3.9	43.5	4.4	42.1	1.5	34.3	2.0	43.0
Sink	8.9	74.6	23.1	75.0	7.8	66.9	13.5	66.3	12.6	65.3	10.0	64.7	11.5	69.3	9.6	69.5
Fines[a]	18.5	14.3	16.4	18.7	12.5	18.3	15.4	19.6	12.9	19.3	11.3	16.6	31.0	11.8	16.5	20.2
Cumulative ash[b]		14.1		24.9		17.5		20.3		19.5		17.5		12.9		16.6

[a] Fines percentage is calculated on basis of total sample weight. Float-sink tests are on plus 20-mesh only, and are calculated to 100 pct of coarse material.
[b] Cumulative ash is for total sample, including fines.

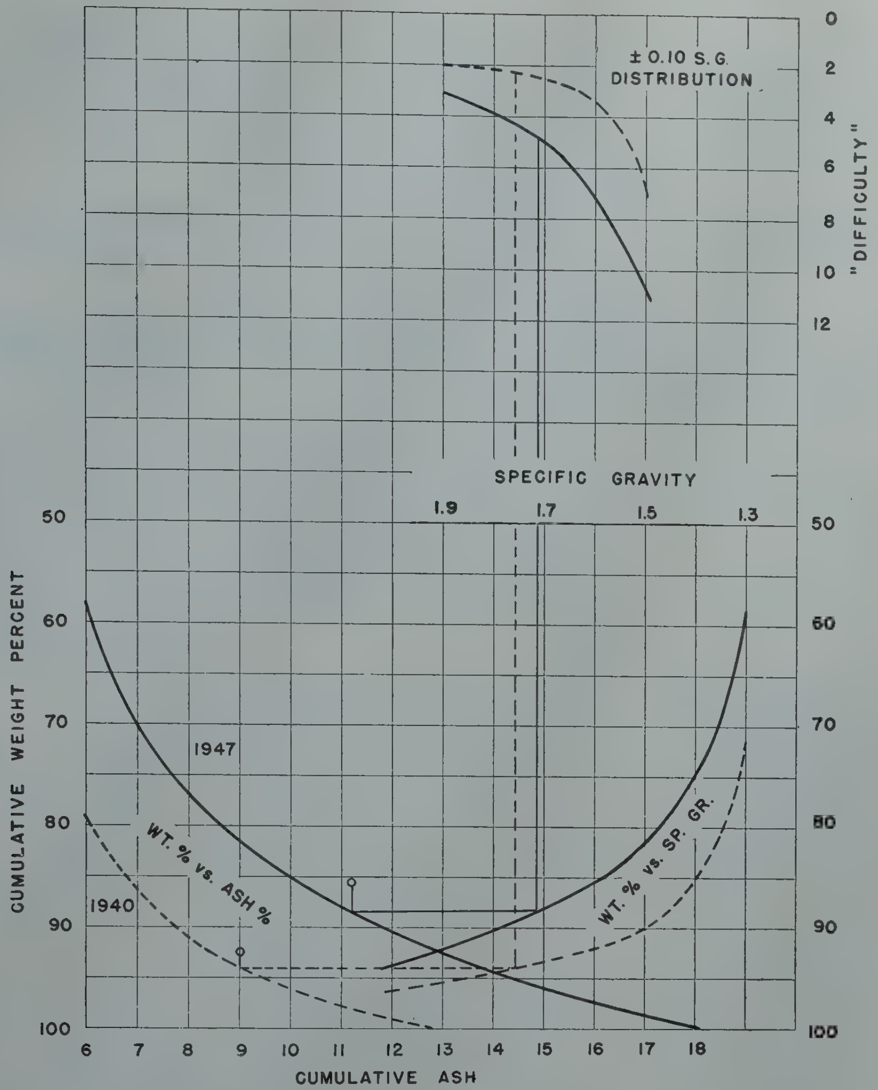

FIG 6—Washability curves, Pueblo coals.

ses of samples taken from the tops of individual cars, according to a regular schedule of proportion of cars of each coal to be sampled and location of sampling points. "General opinion" is that such samples are unreliable; but our experience leads us to believe that this is not necessarily so. The cumulation of all car samples shows an average ash of 17.9 pct as compared to the cumulative ash of 18.1 for all "stream" samples of the mixed feed to the washery. The fines in the two sets of samples are virtually identical at 16.5 pct, and the same is true of volatile matter and sulphur. Any bias in the top samples is, for practical purposes, negligible.

The float-sink data on the average mixture for the year 1947 is shown in Table 16. The usual "separation point" is at approximately 1.7 sp gr, hence the expected ash in the washed coal is slightly more than 11 pct. The cumulation of all washed coal samples shows an ash of 11.2 pct and the yield, corrected to a dry basis, was 85.5 pct of the feed. According to the washability curves of Fig 6 this is an overall plant efficiency of 96.6 pct at a plus or minus 0.10 *difficulty* of 5 pct.

As a matter of interest, we have added to Fig 6 the washability curves for the 1940 coal mixture. It will be noted that the coal was then much easier to wash than it is now. The indicated efficiency was 98.1 pct at a *difficulty* of 2.2 pct.

A part of the decrease in general desirability of the coal mixture is due to the increased usage of the bony coals 4, 5, and 6. However, had the proportion of all the coals in the mixture remained unchanged there would still have been an increase in the difficulty of washing to a low ash: for all coals in use in 1940 have deteriorated noticeably in quality. The total ash has increased considerably, in some cases to as much as 150 pct of the 1940 figure; but this is not entirely due to the inclusion of more "rock"—the float ash at 1.6 sp gr has also increased, with the exception of coal 1.

Table 16 . . . Average Mixed Feed Coal, Float-sink Data for the Year of 1947, Pueblo

Specific Gravity	Weight, Per Cent	Ash, Per Cent	Cumulative Weight	Cumulative Ash
1.30	58.5	6.0	58.5	6.0
1.38	14.0	13.0	72.5	7.3
1.45	6.4	20.6	78.9	8.4
1.50	3.0	27.4	81.9	9.1
1.60	3.6	34.3	85.5	10.2
1.70	2.6	42.7	88.1	11.1
Sink	11.9	70.1	100.0	18.1

The change in washability of the coal has not affected the type of reject

from the tables to any great extent, as is shown in Table 17, comparing the average float-sink of refuse for 1940 and 1947. The quantity of reject is, of course, greater now than in 1940.

Table 17 . . . Comparison of Rejects, 1940 and 1947, Pueblo

	1940		1947	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
Float[a]	4.5	9.9	4.7	11.8
Bone[b]	8.0	27.8	8.5	29.8
Rock[c]	87.5	71.8	86.8	71.6
Head ash		63.8		64.1

[a] Float on 1.38 sp gr.
[b] Float on 1.60 sp gr.
[c] Sink in 1.60 sp gr.

The changes in specific gravity consist of the washed coal are due to two effects in addition to the obvious change in specific gravity consist of the feed shown in Fig 6. The first of these is the sizing action of the coal washing table, which makes it virtually impossible to effect a separation of this coal by specific gravity alone, and the second is the disintegrating action of the centrifugal driers.

As it leaves the table, the washed coal contains more *bone* and less *coal* and *rock* than is indicated by Table 18 (which is based on dried coal). A part of the bone is not homogeneous (as was that at Washery *J*); but is large laminated material which, on further crushing, becomes *coal* and *rock*.

Table 18 . . . Comparison of Washed Coals, 1940 and 1947, Pueblo

	1940		1947	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
Float	90.8	6.7	85.4	7.9
Bone	7.3	24.9	10.5	27.0
Rock	1.9	48.1	4.1	50.1
Head ash		9.0		11.2

The usually recommended solutions in such a case are:

1. Crush the material finer before washing. 2. Take the *bone* out as a middling, crush to liberate the impurity, and rewash. Neither of these schemes is entirely acceptable at Pueblo. Further crushing of the feed before washing would increase the amount of fines, and this has already passed the capacity of the fine coal handling system. Removal of this material as "middling" is out of the question since it reports near the feed end of the table.

Underlying this difficulty is the fact that the low-ash coal is more friable than the bone or rock in practically all coals used at Pueblo. This is illustrated to a certain extent by the screen tests on raw coal, washed coal and reject. In Table 19 we show the averages for the year of 1945, the latest data available.

Repeated zonal tests of table performance indicate that the size (and shape) as well as the specific gravity of the individual particles must be taken into account in attempting to predict the results to be obtained in the table washing of coal. Large material of low specific gravity will cross the riffles much more readily than small material of the same specific gravity.

This can, and at Pueblo does, result in the recovery of large *bone* nearer the feed end of the table than would be expected on the basis of specific gravity alone. If this *bone* is laminated material (and no washer can distinguish between homogeneous *bone* and *laminated coal and rock*), subsequent rough handling will separate the layered components, giving the erroneous impression that the process is unable to separate light rock from coal.

With a greater amount of this laminated material in the feed than in previous years, it is not all surprising that there is an increased amount of *rock* in the washed coal. Regardless of the assignable reasons for such a condition, the presence of rock in washed coal is not desirable: but it does point out the fallacy of assuming that the specific gravity distribution alone is a reliable criterion of the difficulty of obtaining a clean washed coal.

Table 19 . . . Size Consist and Ash Content of Raw Coal and Washery Products, 1945

Screen Size		Raw Coal		Washed Coal		Reject	
Through	On	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
	0.525	9.4	27.6	1.9	17.9	16.5	66.0
0.525	0.371	13.8	20.4	5.4	15.7	16.5	66.0
0.371	0.263	13.0	17.1	7.2	13.9	11.0	71.4
0.263	0.131	21.1	14.9	17.4	11.7	14.7	71.4
0.131	0.065	16.1	13.3	19.8	9.6	13.0	65.1
0.065	0.033	11.2	13.7	19.4	8.1	11.8	58.8
0.033		15.4	16.4	28.9	9.2	18.5	58.2

How Do the Various Washing Systems Compare?

An incomplete comparison of the products of the various types of washers shows the Baum jig as the best, followed in order by the diaphragm jig, the table, the basket jig, and the plunger jig, respectively (Table 12). The comparison is based on the proportion of obviously misplaced material in the washed coal and reject. As was just shown, this is by no means the whole story; but it is probably as good a basis as can be found in the limited data available.

Table 20 . . . Comparison of Out-of-place Material in Products from Various Types of Washers. Colorado-New Mexico Practice, 1947 to 1948

Type of Washer	"Rock" in Washed Coal, Per Cent	"Coal" in Reject, Per Cent	Total, Per Cent
Baum jig	1.1	2.3	3.4
Diaphragm jig	1.8	3.7	5.5
Table (wet)	4.1	4.7	8.8
Basket jig	1.9	9.5	11.4
Plunger jig	7.8	8.8	16.6

Conclusion

Each washery has its own particular set of operating conditions and problems—and not the least of these is satisfying the customer. Operations which, at first glance, appear to be "wasteful" or "unscientific" actually may be economically sound for a particular set of conditions. As conditions change, the operator will have no choice but to change his practice or to drop behind those who do. Indications are that the operations covered in this survey will be improved as necessity dictates.

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1. Marius R. Campbell: Our Coal Supply. Int. Conf. on Bit. Coal, 1926.
2. Coke Plant Laboratory. Colo. Fuel and Iron Corp., Pueblo, Colo.

The Rupp-Frantz Vibrating Filter[1]

W. M. BERTHOLF* and J. D. PRICE,* Members AIME

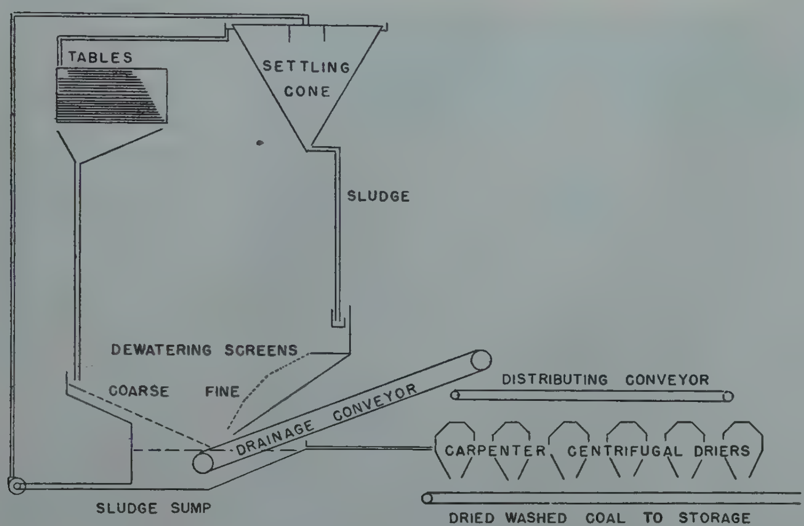

FIG 1—Original layout of washery showing possibility of building up recirculation of fines (sludge).

One of the chief difficulties with which the operator of a coal washing plant has been forced to contend is the handling of the very fine coal. First he has the problem of separating the fine coal from washery water. This is usually accomplished by the use of settling cones or Dorr thickeners: in either event the separated fine coal contains a high amount (40 to 80 pct) of water and, somewhat loosely, dependent upon the size of the coal particles present, may be known as sludge or slurry. His second problem is the satisfactory further dewatering of such separated sludge and/or slurry.

At the coal washing plant of the Colorado Fuel and Iron Corp., washed coal from the Deister Plat-O tables is sluiced to a stationary inclined dewatering screen provided with 3⁄16 by 1-in. slots, and the coarse coal passing over this screen is conveyed by a Baum-type elevator and a flight-type distributing conveyor to a battery of Carpenter centrifugal driers. The fine coal and water passing through the openings of the primary dewatering screens, plus the fine coal passing through the screens of the centrifugal driers (and no small portion of which is the result of the disintegrating action of these driers) is pumped to two 30-ft Link Belt settling cones. The sludge recovered as underflow from these cones was formerly dewatered on stationary screens having 1⁄16 by 1-in. slots and discharged on top of the coarse washed coal in the Baum drainage conveyor; but after a relatively few hours of operation the circulating load of fine coal on this conveyor built up to a point where it became necessary to reduce the quantity of feed to the washing tables. The original arrangement of our coal washing and dewatering equipment is shown in Fig 1.

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TP 2549 F. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before June 30, 1949. Manuscript received Nov. 1, 1948.
* Efficiency Engineer and Superintendent, respectively, By-Product Coke Plant, Colorado Fuel and Iron Corp., Pueblo, Colo.
[1] References are at the end of the paper.

In order to regain the loss in active throughput which had been caused by this circulating load, an installation of six vibrating filters was made during the year of 1936. This resulted in an increase in normal tonnage of washed coal from 1430 to 1750 tons per 8-hr shift, a gain of slightly over 22 pct. This increase in throughput has made a very substantial decrease in the "conversion cost" per ton of washed coal. Since the vibrators were installed, we have gained about 2,000,000 tons in production, compared to what would have been produced in the same number of shifts at the former rate. Putting it in slightly different form, we have saved in the past 12 yr the cost of 1400 operating shifts in the washery.

Fig 2 shows the location of the vibrating filters as applied to the original flowsheet. It will be noted that the sludge from the bottom of the settling cones is passed through a distributing box and thence to the filters. The dewatered coal from the filters is delivered to the conveyor receiving the dried coal from the Carpenter driers.

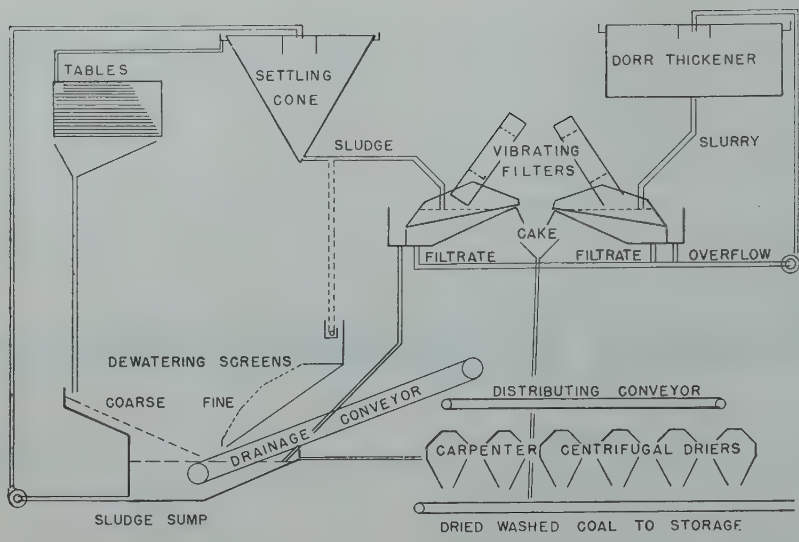


FIG 2—Location of vibrating filters in washery flowsheet. Five vibrators handle sludge, one handles slurry.

up plate and a screen wire cloth.

3. An electrical vibrating unit.

The frame proper needs little additional description, beyond the fact that it is made up of 3/16-in. steel plate, with careful attention being paid to the welding to ensure against failure due to the heavy vibration to which the entire filter is subjected. The frame is supported, or hung, directly from the vibrating unit, this being attached to the frame by the cross members which are securely bolted to the "bull head" of the vibrator.

The screen assembly, shown in Fig 5 and 6, is made up of a foundation composed of a rectangular frame of plate and angles with special stiffening cross members, a backing-up plate and the filter screen proper. Fig 5 is a bottom view of the completed screen assembly. Note that there are no intermediate longitudinal members. The transverse stiffeners are unequal leg angles with an included angle of 135°, the width of the long leg being 2⅝ in. at the edge of the frame and 5 in. at the center. These stiffeners are arc welded to the side members of the foundation, which are bolted into the main frame. The backing-up plate, a bronze plate of 8-gauge material and provided with ⅛ by ½ in. perforations, lies directly on the structural members of the foundation and is covered with a Monel metal screen, 35-mesh, with 0.015 in. wire. All three sections are rigidly bolted together with 5/16 by ¾ in. galvanized carriage bolts spaced on 3½ in. centers across the horizontal leg

By this system the sludge bypasses the drainage and distributing conveyors and the centrifugal driers and the circulating load is minimized. At such times as the filters are able to handle the entire load, there is no material put over the secondary dewatering screens.

Three separate products are taken from the filters, as follows:

1. The filter cake, representing the dried and recovered portion of the feed which, as has been shown, is added to the dried coal leaving the Carpenter driers.

2. An overflow product, which is a "safety valve" discharge from an overflow gate on the filter, provided to prevent overloading of the filter and consequent poor operation. This overflow product is returned to the main system and again pumped to the settling cones, in the case of the five vibrators operating on *sludge*. The overflow from the *slurry* vibrator is returned to the Dorr thickener.

3. The filtrate, or material passing through the screens of the filters, which is pumped to a Dorr thickener on the fourth floor of the vibrating filter structure and from which the settled slurry flows by gravity to a separate but similar filter on the second floor of this structure. Fig 3 shows the arrangement of equipment in this section of the building. Note that the slurry could be added to the total flow of sludge, added to the sludge fed to a single vibrator, or treated separately. We use the last method unless the vibrator in this slurry service is out of commission, and then revert to adding the slurry to the distributor, from which it flows to all vibrators.

The Filter Mechanism

We now come to the details of construction of the vibrating filter and these are shown in Fig 4 to 7, inclusive. It will be noted that the filter proper consists of three main parts:

1. A frame of welded and riveted construction (Fig 4) to support the screen assembly.

2. A screen assembly (Fig 5 and 6) consisting of a foundation, a backing-

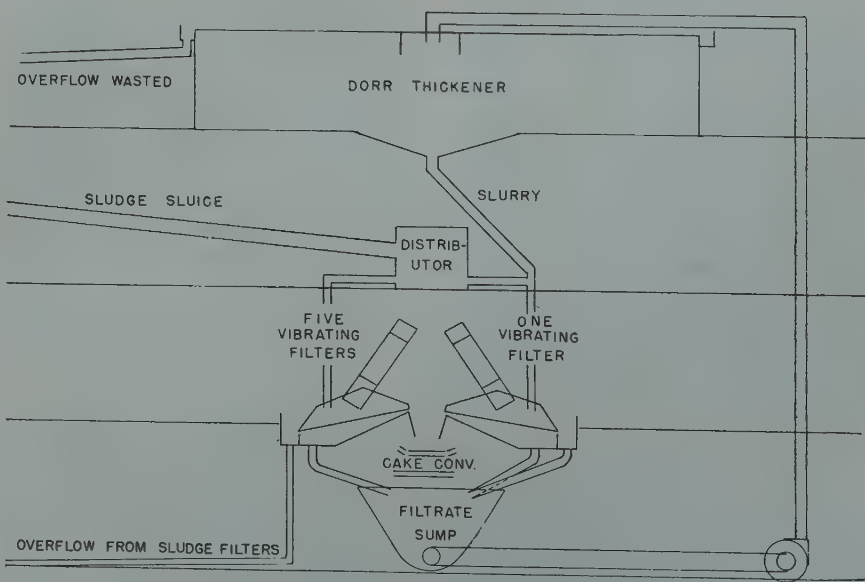
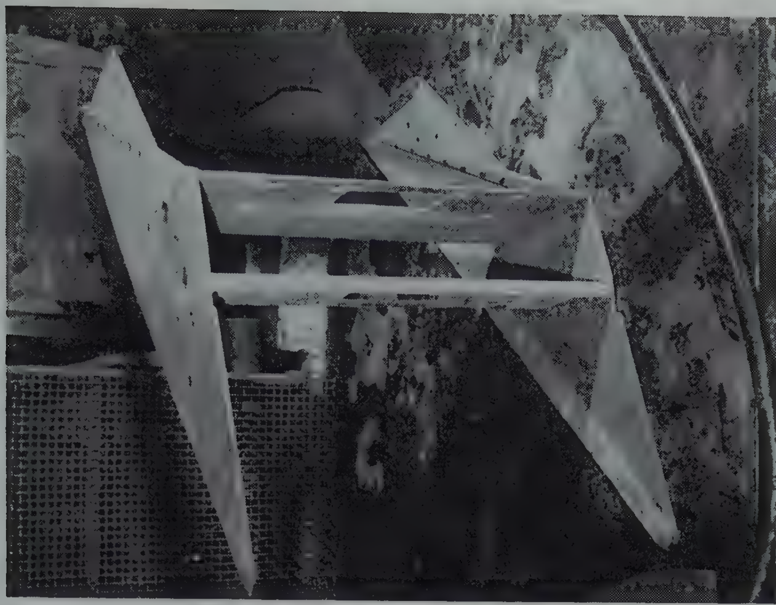


FIG 3—Location of related equipment on four floors of filter structure.



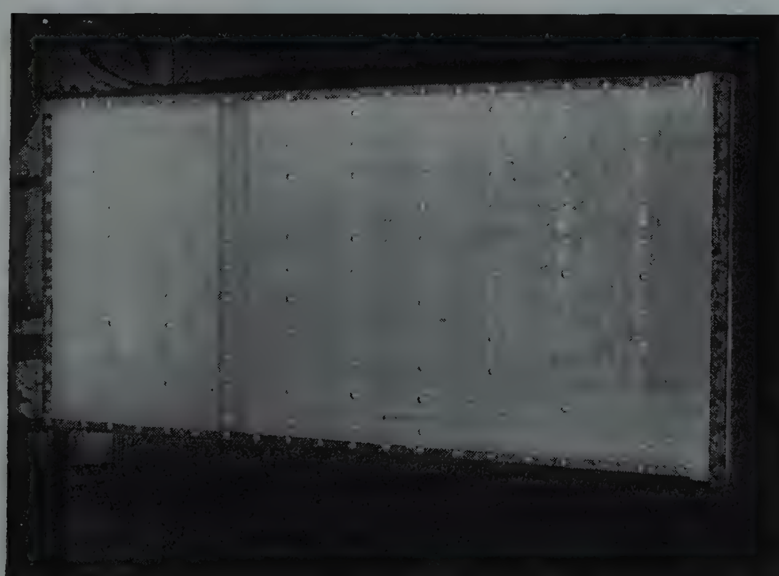


FIG 4—Filter frame. (*Top.*)
FIG 5—Bottom view of completed screen assembly. (*Center.*)
FIG 6—Top view of completed screen assembly. (*Bottom.*)

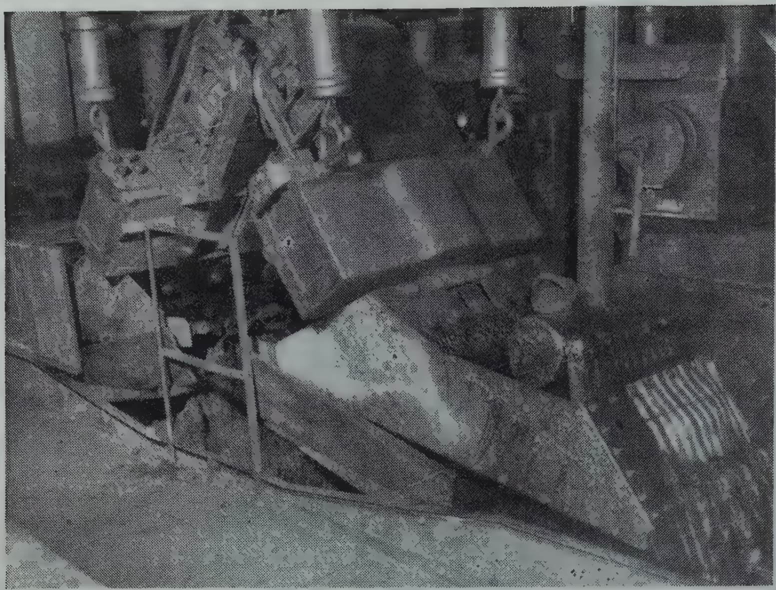


FIG 7—Filter in operation, view from feed end. (*Top.*)

FIG 8—Filter in operation, view from discharge end. (*Bottom.*)

of each stiffening angle and along the side members of the foundation. Fig 6 shows a top view of the completed screen assembly. The vibrating mechanism is a standard Jeffrey-Traylor unit, No. 5, operated on mixed ac-dc current and rated at 30 amps at 220 v. Fig 7 is a view of the complete assembly from the feed (overflow) end. The complete unit is suspended from the floor above through standard spring vibration absorbing hangers.

While it may appear that the arrangement is cumbersome and that more attention has been paid to detail than should be necessary, in reality every point of design has been arranged for some particular purpose. The slope of the stiffening angles and the large number of assembly bolts are for the specific purpose of preventing secondary vibrations. To secure satisfactory results every part, every square inch, of the complete assembly must vibrate in absolute unison. If any secondary vibrations or harmonics are set up, the machine will fail to operate satisfactorily. Drum-head tension of screen cloth or other characteristic construction details found in many vibrating screens would make impossible satisfactory dewatering of the sludge.

The Filter in Action

Now, let us examine the machine in action. In Fig 7 the sludge is being fed into the vibrator through a pipe and distributing trough. The assembly is so supported that, when under load, the screen bed will incline upward at an angle of about 7° from the horizontal. This arrangement will permit the formation of a pool of sludge up to the overflow gate on the back of the frame and extending about three-fourths of the way up the slope of the screen. In Fig 7 we see the excess feed being discharged through the overflow gate, which is covered by a rubber flap.

When vibration is applied to the machine, things begin to happen. The sludge settles to the bottom of the pool, the vibrating action packs the sludge and squeezes out water, the compacted cake then starts moving up the screen, due to the differential stroke employed, and is compacted further in the process, more and more of the excess water being squeezed out. In the final stage, the cake moves out of the pool, dewatering is completed, and the "dry" cake is discharged as shown in Fig 8, a view from the discharge end. This cake has had its water content reduced to within 2 or 3 pct of that which would be obtained with a vacuum filter. The cake at point of discharge is dry enough and compact enough so that when it extends some distance beyond the end of the screen it will break off sharply because of its weight. If the cake bends to any pronounced extent before breaking off, the operator knows that the unit is not functioning properly. A normal cake is apparently dry and can be handled without wetting one's hands, whereas a poor cake will contain some visible excess water.

The water separated from the sludge is split between overflow and filtrate. Ideal operation would, of course, involve no overflow; but this requires the use of a feed which is difficult to obtain and difficult to handle once it is obtained. We would prefer to operate with a feed containing between 50 and 60 pct solids. Such a feed would eliminate the necessity for overflow and would almost double the tonnage with the same dewatering effect as we now obtain with feeds containing 35 to 45 pct solids. The greater part of the filtrate goes through the lower end of the screen, where there is a considerable depth of pool, and where compaction of the cake has not been fully accomplished.

Some Fundamental Considerations

All of this sounds fairly simple; but there are a number of factors which contribute to its success, the failure of any of which will cause unsatisfactory operation of the machine.

1. The machine must be so constructed and maintained that all parts vibrate in unison. Only by such vibration is the filter cake formed, and it must move as a unit. If for any reason "ribbons," "snakes," or "balls" of sludge are formed, the dewatering action is impaired and the capacity of the machine is seriously reduced. Any secondary vibration, especially between the wire screen cloth and the backing-up plate, will reduce the life of the parts.

2. The coal in the sludge must have such a range or ratio of sizes that it will form a compact cake under the influence of applied vibration. This machine will not function properly on coarse coal or on sludge if there is a serious deficiency in the finer sizes. The situation may be compared to the problem of making a satisfactory concrete. The proportions of gravel, sand, and cement must be properly balanced for good results. While much remains to be learned about the requirements for this particular machine, it appears that material conforming rather closely to the Rosin-Rammler law of size distribution, with a fairly large range of sizes, is capable of being treated with satisfactory results. It certainly is not necessary that the material conform to the theoretical requirements for maximum density of packing which (when dealing with spherical particles) can be obtained only when the size distribution is discontinuous, that is, with components of large, small, and very small sizes but no intermediate gradations.

3. The applied vibration must be of the proper characteristics. Frequency is substantially constant for a given installation, depending upon the alternating current component of the mixed current. Wave form is determined by the relationships between spring-bar tension, the total sprung weight of the mechanism under load, and the wave form of the mixed current. It is the resultant of the magnetic, elastic, and gravitational forces involved. The amplitude is also a resultant of these forces; but in practice is controlled by adjustment of the flow of current to match the load being carried. Too little current will result in a stalled cake and too much current will result in carrying an excess of water up the screen with the coal.

4. The sludge or slurry fed to the vibrator should contain a reasonable percentage of solids, for the capacity of the filter in terms of dry cake discharged per hour is dependent entirely upon the amount of work the filter must do: and this is (apparently) measured in terms of water to be removed. The smaller the proportion of water which must be handled, the greater the amount of dried cake which can be delivered.

If these factors are kept reasonably constant and within reasonable limits, very satisfactory results can be secured.

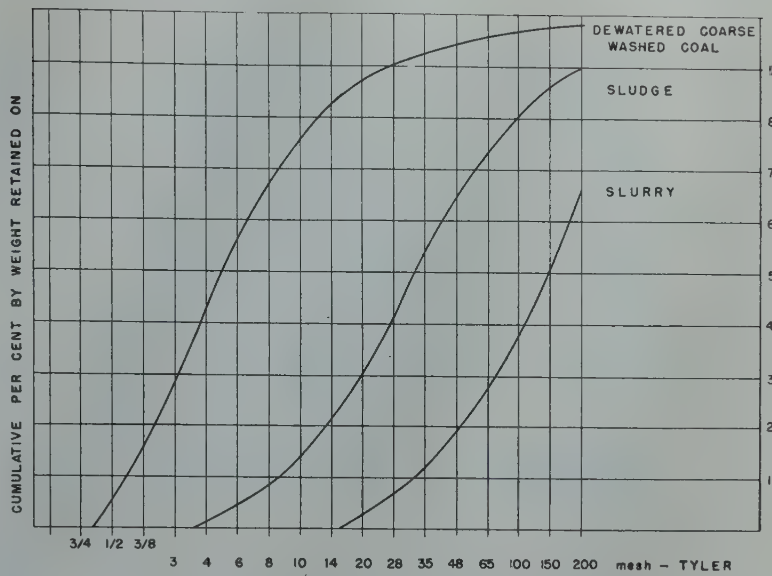


FIG 9—Comparison of various fractions of washed coal.

Operating Results

Results are, of course, the "proof of the pudding" and may be considered from four angles: (1) capacity of the machine, (2) dewatering effect, (3) distribution of sizes among products, and (4) cost of operation. Each of these appears worthy of separate discussion.

CAPACITY OF THE MACHINE

As has been pointed out above, the capacity of the machine is limited by the amount of water in the sludge. It is also affected by the fineness of the solids. The area of the screening surface and the size of openings in the screen cloth are fairly well established and can be modified only within rather narrow limits. The maximum size of screen which we have been able to operate satisfactorily is 3 ft wide by 6 ft long. Above this size the extra weight of material which would be necessary to ensure absolute rigidity is so great that the useful load is reduced. With the 3 by 6 ft screen, and with a feed containing 40 pct solids, a cake production of 10 tons per hour per unit is regularly secured. If the feed is prethickened to 50 pct solids, the capacity is increased to 15 tons and with a feed prethickened to 60 pct solids, to 20 tons per hour. Since the cake with a normal sludge feed (practically all minus 6 mesh) will contain 20 to 24 pct water, the above capacity figures must be reduced by about 22 pct or to 7.8, 11.7 and 15.6 tons of *dry* fine coal per hour.

The screen cloth must have a certain amount of sturdiness in its own right: it is not possible to use a flimsy screen cloth, even with the large number of bolts used in the assembly. If the screen cloth is too fine or has a very limited percentage of open area, it may not be possible to completely dewater the cake by the time it reaches the end of the filter bed. If the cloth is too coarse, there is an appreciable increase in the fine coal in the filtrate. We have obtained better results with 35 or 40-mesh screen than with any others tried; but the wire must be of sufficient size to make a sturdy cloth and must not be too brittle.

DEWATERING EFFECT

The amount of dried sludge cake delivered by the unit has little, if any, effect upon the moisture content of the cake, and, at an operating rate of 20

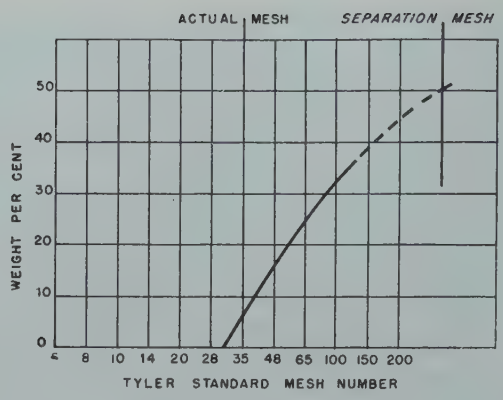


FIG 10—Percentage of available material (cake plus filtrate) passing through the screen when drying sludge. The filter is not an efficient screening device.

tons cake per hour, cake moisture has been recorded lower than with 10 tons output. However, the particle size of the material present in the sludge or slurry does have an important bearing on the moisture content of the final product.

At the Pueblo plant we are dewatering two materials with this equipment, the settling cone underflow, which we call sludge, and the Dorr thickener underflow, which is called slurry. It has already been mentioned that the sludge will nearly all pass a 6-mesh screen. Since the Dorr thickener material is made up solely of the filtrate which passes through the vibrating filter screens, this material is all under 35 mesh in size. The screen analyses of cakes made from these two products is plotted on Tyler standard logarithmic paper in Fig 9, where we have also shown, for comparison, the coarse dewatered coal from the primary dewatering screens. It will be noted that these three materials are widely separated in size and differ appreciably in the proportion of extremely fine material. With so few fines in the dewatered coarse coal, it is easy to understand why there would be difficulty in producing a coke which has most of the water *squeezed* out of it. Many trials were made with this type of material; but it was impossible to obtain a satisfactory cake, and the dewatering action was limited to shaking excess water off the coarsest pieces. Material approximating the screen analysis shown for the sludge cake will form a true cake, from which the greater part of the water will be squeezed out, as will the finer material from the Dorr thickener. As has been noted above, and by others,[2] the retained moisture is affected considerably by the particle size distribution, the greater the amount of fines the greater the amount of moisture retained. For this reason, it is not possible to reduce the moisture content of the slurry cake to the 20 to 24 pct figure quoted for the sludge, its moisture usually being 4 to 5 pct higher. However, a good cake from the slurry vibrator also appears dry on visual examination and may be handled without actually wetting the hands, indicating that practically all excess water has been removed. The capacity of the slurry vibrator is much lower than the sludge vibrators, being approximately one third as great.

DISTRIBUTION OF SIZES AMONG THE VARIOUS PRODUCTS

Here it may be opportune to state that the filter should not be considered as a screen. In fact, it is about as inefficient a screen as might be devised, as is shown in Fig 10, where it will be noted that the actual screen is 35 mesh whereas the 50 pct separation point is in the neighborhood of 270 mesh, that is, half the 270-mesh material remains in the cake and half passes the screen as filtrate. The data for Fig 10 were secured from a test on sludge; but the situation is similar when drying slurry.

The general qualitative situation is shown in Fig 11, where it will be noted that the cake is the coarsest product and the feed, overflow, and filtrate are progressively finer. It is worth noting that the overflow does not carry the very largest particles in the feed—they are able to settle in spite of the rapid current; hence the cumulative curve for the overflow starts at a point to the right of the maximum particle size present in the feed. The cake curve is "deformed" and rises above the feed curve, even though both start at the same point. Normally the filtrate can contain nothing larger than the nominal mesh of the screen, hence its curve should start at about 35 mesh. The presence of oversize is an indication that the screen is torn.

Fig 12 shows the distribution of the various sizes in the feed both as a percentage of the total solid feed and as distributed among the products in a manner similar to the familiar Han-

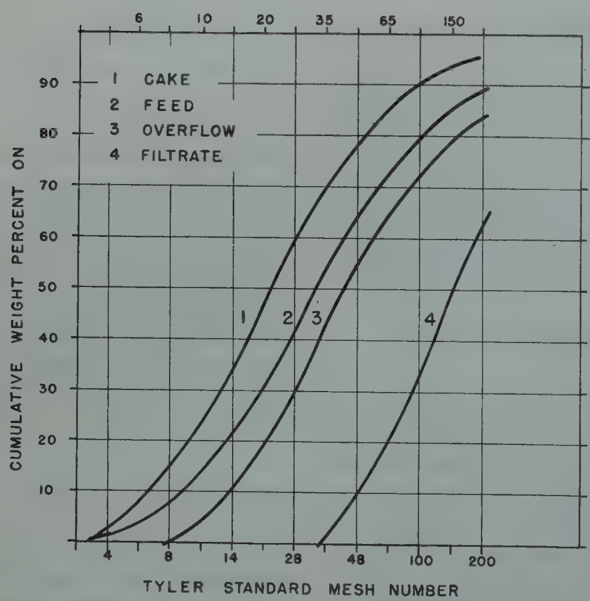


FIG 11—Typical screen analyses of filter products.

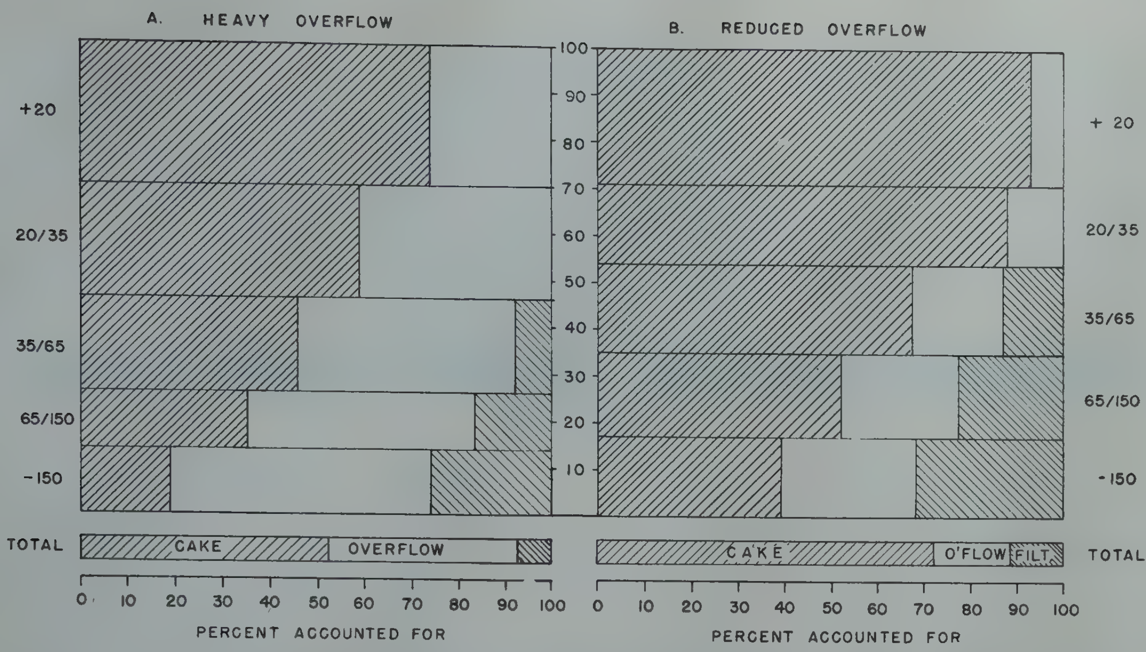


FIG 12—Distribution of various sizes in feed among filter products.

cock diagram for distribution of ash in coal. Two conditions are shown, (*a*) operation with a heavy overflow, and (*b*) operation with a reduced overflow. The reduced overflow was secured by reducing the feed until only a small amount of overflow persisted; but not sufficient to reduce the cake tonnage appreciably. The reduction in the feed rate was about 42 pct and the reduction in the cake tonnage was about 20 pct. There are slight differences in the size composition of the two feeds; but we do not believe they are sufficient to cause any noteworthy difference in behavior. For the two tests there is very little difference in the net recovery of "available" solids, the percentage of such solids in test *a* being 88 and in test *b* being 87. In both tests there is a complete recovery of material coarser than the screen, and a decreasing recovery as the size decreases.

The provision for overflow is intended to prevent the building up of an excessive load in the lower end of the filter, a condition which could result from surges in the feed rate. In practice it has a legitimate function in permitting the operator to feed the filter at a rate slightly above normal capacity, the excess being recirculated and the "net feed" remaining constant at an acceptable rate. Operation along these lines is satisfactory; but when the operator believes that the overflow is intended to save him "trouble" (paying attention to what is going on) it is possible to over-do a good thing. He can then open the feed cock and let the vibrator take care of itself the remainder of the shift. Since a thin sludge will flow much more readily than a thick sludge, there is a tendency to feed the vibrators "too much and too thin"; and the results are not up to standard.

COST OF OPERATION

It will be recalled that in describing the construction details of the filter it was pointed out that a bronze backing-up plate and a Monel woven wire screen were used. These have been adopted after trials of many other materials and have shown longest tonnage life and lowest cost per ton. A further reason for the choice of these materials is found in the necessity of using nonrusting metals: steel has been found unsatisfactory. It should also be mentioned that, in order to retard rusting action, the screen foundation and the fastening bolts are galvanized. Emphasis must be placed upon the need of frequent inspection and prompt repair if maintenance costs are to be kept low. This is particularly true of rips or tears in the screen cloth proper: if neglected, coal will work under the screen and greatly shorten its life.

Based on operating experience on six filters over a period of more than ten years, average life of parts and cost of operation are shown in Tables 1 and 2.

Table 1 . . . Useful Life and Cost of Filter Components

Item	Life, Tons	Initial Cost	Cost per Ton
Main frame	100,000	$25.00	$0.0003
Screen foundation	40,000	50.00	0.0013
Backing-up plate	250,000	75.00	0.0003
Screen cloth	15,000	50.00	0.0033
Total replacement material cost, per ton cake			$0.0052
Current maintenance material cost, per ton cake			0.0028
Total maintenance and replacement material cost			$0.0080

Table 2 . . . Operating Cost per Ton of Dry Coal Handled

Operating labor	$0.0220
Supervision	0.0055
Maintenance material	0.0080
Maintenance labor	0.0100
Electric power and light	0.0020
Total operating cost	$0.0475

Conclusion

The Rupp-Frantz vibrating filter is not an ordinary dewatering screen, many types of which have been tried in this same service with indifferent results. The design and operating features which we have stressed are fundamental, and when all requirements are met, this piece of equipment fills a long-recognized vacancy in the field of coal washing apparatus.

References

1. U. S. Patents 2089548 (1937) and 2183896 (1939).
2. J. M. DallaValle: Micromeritics. Chap. 11, pp. 213–222. Pitman.

Synthetic Liquid Fuels from Coal

By J. D. DOHERTY,* Member AIME

That America's great coal deposits eventually will be our principal source of liquid as well as solid fuels is generally accepted. Moreover, the day when synthetic oil from coal will begin to supplement our petroleum supplies does not appear to be far off.

The American petroleum industry has not spared effort or expenditure in its attempt to keep pace with the demand for oil, and recent production has exceeded expectations. However, the rapidly rising demand for liquid fuels plus the increasing difficulty of finding and developing the new oil needed even to maintain present production rates make it highly unlikely that oil from domestic wells can fully meet future oil requirements. Therefore, it appears that one or more of the following courses must be followed:

1. Curtail oil consumption.
2. Incr ase imports.
3. Develop new sources such as synthetic liquid fuels.

All three courses may well be needed, as it is doubtful that any one of the three alternatives alone can satisfactorily solve the problem indicated by the prevailing trend in demand for oil.

Curtailment of oil consumption is not a constructive policy, except where oil is being wasted or coal can be readily substituted as in stationary steam plants. Reduced oil supplies for most uses will curb our economic advancement and, in addition to inconveniencing consumers, will have a pronounced adverse effect on American industry. The oil-burner-manufacturing industry already has been dealt a hard blow. Shipments in the first 5 months of 1948 decreased 70 pct from those in the same period in 1947.[1] The effect of a similar cut in the automobile industry can be imagined. A severe prolonged shortage of gasoline could seriously disturb our present economic well-being. It might be pointed out also that even the prospect of a shortage or lack of assurance for the future can discourage buyers of equipment, as occurred in the case of oil burners.

An increase in our imports of oil seems to be a necessary expedient for the immediate future. Its advantages are that it will require a minimum of manpower and steel in a time of critical shortages and will conserve domestic reserves. Any major dependence upon foreign sources for our oil supplies should be avoided, however. Other nations are increasing their oil consumption more rapidly percentagewise than the United States and eventually will compete with us for the not unlimited world supplies. In peacetime, foreign countries might restrict shipments of oil to this country or impose exhorbitant prices. In the event of war, there is serious danger that such sources would be cut off, and American-developed oil fields overseas might even be used against us.

Although synthetic fuels cannot relieve any oil shortage that might develop within the next year or two, an immediate start toward a substantial synthetics industry to supplement petroleum not only could help relieve shortages several years hence but also would help establish the much needed "know-how" for large-scale production. By assuring ample future supplies, such a start would permit sound planning by industries dependent upon oil and maintain or restore the confidence of prospective customers for oil-consuming equipment.

If it is agreed that domestic production of crude oil cannot be expected to meet future domestic demands, that oil rationing in general is undesirable, and that too great dependence on foreign oil should be avoided—and the weight of the evidence supports such conclusions—it is inescapable that commercial synthetic fuel production should be developed to supplement domestic petroleum supplies, which, of course, will furnish the bulk of our requirements for years to come.

Sources of Synthetic Oil

The three principal sources of synthetic oil are natural gas, oil shale, and coal. A commercial plant using natural gas is under construction, and expansion of this industry is expected.

White Sulphur Springs, November 1948.

Paper presented at Joint Fuels Conference of Coal Division AIME and Fuels Division ASME.

TP 2562 F. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before June 30, 1949. Manuscript received Sept. 27, 1948.

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[1] References are at the end of the paper.

However, as the Nation's natural-gas reserves are not much larger than its reserves of crude oil on a basis of weight or heating value, most of the natural gas is committed to pipelines, demands for fuel gas are increasing, and the number of Btu's in fuel of high form value that can be delivered through gas pipelines is about twice that which can be delivered in liquid fuel made from the gas, it is not anticipated that more than a few percent of our gasoline requirements ever will come from this source.

United States oil-shale reserves contain recoverable oil equivalent to at least 125 times our present annual consumption; and, partly because of the low mining costs being demonstrated by the Federal Bureau of Mines at Rifle, Colo., this alternate source appears to be the most promising economically for large-scale oil production. Large as these deposits appear, however, they are small when compared with the country's coal reserves. Furthermore, economically favorable deposits of shale are concentrated in Colorado, Utah, and Wyoming, some distance from the largest markets, and shale is best adapted to the production of heavy fuel oil, hydrogenation or special refining being required for large yields of motor fuel. For these reasons the production of Diesel oil and gasoline from coal is likely to be developed simultaneously with shale oil.

Coal is by far the leading potential source of oil in the United States. Fig 1 indicates the quantities of oil that can be produced from estimated United States reserves of petroleum, natural gas, oil shale, and coal. The accuracy of the geologic estimates of coal reserves has been questioned, and they are being revised constantly. However, a glance at the chart will show that no reasonable revision could alter the conclusions that coal is our outstanding fuel reserve and that coal can furnish oil at present rates of consumption (2 billion barrels per year) for centuries to come.

Suggested Program

To meet anticipated increases in domestic consumption of liquid fuel, it has been suggested that plants be built within the next 10 years to produce 2 million barrels of synthetic oils per day from coal, oil shale, and natural gas. Assuming that half of this production, or 1 million barrels per day, would be produced from coal, the effect of such a coal-to-oil program can be estimated. For the purpose of this paper it will be assumed further that oil from coal will be produced equally by the gas synthesis (Fischer-Tropsch) and coal hydrogenation processes and that half of this production will be from coal east of the Mississippi River and half from coal and lignite west of the Mississippi River.

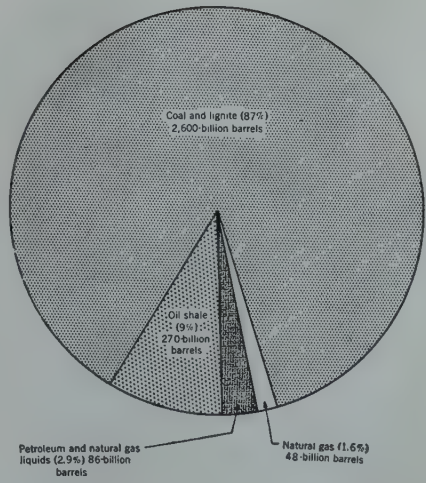

FIG 1—Estimated proved, indicated, and inferred liquid fuel resources of the United States.

The following factors were used for conversion: *Coal and Lignite*—Tonnage of bituminous coal (13,000 Btu per lb) equivalent to total estimated reserves as given by Fieldner (*Mech. Eng.*, March 1947, p. 221); recovery in mining, 50 pct; liquid fuel equivalent—1 ton bituminous coal equals 2 bbl liquid products. *Oil Shale*—Belser's estimate of assay yield of oil in place in Colorado (Amer. Inst. Min. and Met. Eng. TP 2358, *Petr. Tech.*, May 1948) reduced 25 pct for mining losses, and Winchester's estimates of recoverable oil in other States (Append. I of Fed. Oil Conservation Board Report II, 1928). *Petroleum and Natural Gas Liquids*—Based on L. G. Weeks' "Ultimate Petroleum Reserve" estimate of 110 billion barrels of crude oil for the United States (*Bull. Amer. Assn. Petr. Geol.*, June 1948, p. 1094), less 35 billion barrels produced to date, plus 11 billion barrels estimated ultimate remaining potential of natural gas liquids. *Natural Gas*—Calculated on the assumption that the remaining ultimate potential reserves of natural gas are in the same relation to A.G.A. proved reserves of natural gas as the estimated remaining ultimate potential reserves of oil computed from Weeks' estimates are to the A.P.I. proved oil reserves; liquid fuel equivalent—12,000 cu ft equals 1 bbl liquid products.

Other uses of coal and natural gas and refining losses of petroleum and oil shale have been neglected.

COAL REQUIREMENTS

Bureau of Mines estimates that, for this oil production, 575,000 tons of coal and lignite would be required per calendar day, or a daily output of 805,000 tons when operating mines 5 days per week. This amounts to 210 million tons per year, a 34 pct increase over 1947 production. Of this consumption, 213,000 tons per day would be bituminous coal mined east of the Mississippi River and 362,000 tons would be coal and lignite mined in the west. Some of the western coal and lignite might be obtained by expanding present mining operations, but most of it could be obtained only by opening new mines. Examination of the coal-production statistics indicates that, east of the Mississippi River, existing coal mining capacity could produce most if not all of the coal required by additional operating days and by continued mechanization, which is increasing the output per man-shift. This increased production naturally would result in more rapid exhaustion of reserves in existing mines and necessitate opening new mines at an increased rate. However, it appears that a large synthetic fuel industry could be established east of the Mississippi River without opening new mines or employing additional miners, thus saving construction materials that would otherwise be required for mines and employee housing.

CAPTIVE VS. COMMERCIAL COAL

If the synthetic fuel industry were established entirely with captive coal mines—the plants taking the full production of the newly opened mines—it might have little or no effect on the commercial coal industry. Synthetic fuels operations would provide a market for mining machinery and employment opportunities for mining men, but if no coal were purchased or sold, it would have no more effect on coal trade than the captive mines of the U. S. Steel Corp. and its subsidiaries.

On the other hand, if the synthetic industry is established on the basis of purchasing coal from existing mines wherever possible, the coal industry would be provided with a desirable new market for its product. Synthetic fuel plants as now conceived could use almost any size of coal and would consume it at a uniform rate throughout the year. In practice the requirement of uniform quality and the need for low transportation charges might restrict the purchases of any one plant to a small number of nearby mines.

Legislation to encourage synthetic liquid fuel plants by amending the Internal Revenue Code, introduced in the last session of Congress (S. 2372, S. 2373, H.R. 6008, and H.R. 6495, 80th Congress), would strongly encourage captive production for synthetic fuel plants, as such legislation, if approved, would permit accelerated amortization on all facilities acquired for producing synthetic fuels and would permit an increased depletion allowance on coal mined for this purpose.

CAPITAL INVESTMENT

For producing liquid fuels (90 pct gasoline) by the gas synthesis process, it is estimated that a capital investment of $8500 per daily barrel would be required. Coal hydrogenation would require an investment of $9000 per daily barrel for a plant to produce equal parts of gasoline and distillate fuel oil. For greater gasoline production, the capital investment would be only slightly higher. (A plant to produce chiefly fuel oil from oil shale can be erected for an investment of only $3000 per daily barrel.) These estimates include the investment required for mines and for product pipelines where the plants are away from markets large enough to absorb the products. To produce the proposed 1 million barrels per day from coal and lignite would require a total investment of 8.7 billion dollars. This is a tremendous amount of money, but it can be compared with the capital investment of 7.5 billion dollars that the petroleum industry anticipated would be spent on domestic production, transportation, refining, and marketing facilities in the 5 years from 1947 to 1951, chiefly to maintain present production rates with an expected increase during the period of only a few hundred thousand barrels a day.[2] Actually, about 2¼ billion was expended by the industry for new facilities in 1947 alone.[3]

STEEL REQUIREMENTS

An objection raised against the erection of commercial synthetic fuel plants is the large tonnage of steel required. Recent estimates show that, actually, the steel required for new liquid-fuel capacity from oil shale may be less and from coal little if any greater than that required for increased production of petroleum. It has been estimated that 6 to 7 tons of steel are required for each daily barrel of new petroleum capacity, including production and refining.[4,5] The estimated steel required for synthetic gasoline from coal is 6.4 tons per daily barrel for all facilities, including pipelines to markets. Oil-shale plants to produce chiefly fuel oil have the lowest steel requirement—only 3.4 tons per daily barrel.

The steel required to produce 1 million barrels of liquid fuel per day from coal and lignite, assuming that coal east of the Mississippi would be obtained from existing mines, is estimated at 6,300,000 tons.

It should be noted that a program such as that suggested would not require appreciable quantities of steel during the first two years and that the peak demand would come 5 or 6 years after starting the program.

OPERATING FORCE

The assumed production of 1 million barrels per day of synthetic fuels from coal would require a total operating force of 160,000 men, including all miners. The number of miners required would be 20,000 west of the Mississippi and 33,000 east of the Mississippi. To the extent that coal is obtainable from existing mines east of the Mississippi, new miners would not have to be trained.

PRODUCTS

Coal-to-oil plants for which the foregoing estimates were made would produce daily the following quantities of standard products, which could be distributed through existing channels and which would meet current specifications:

	Barrels Per Day
Liquefied propane and butane (L.P.G.)	86,000
High-grade motor gasoline	648,000
Diesel oil and furnace oil	266,000
Total	1,000,000

It will be noted that this program does not provide for any heavy fuel oil. It is anticipated that fuel oil can be obtained more cheaply from shale and that the shale oil will be distributed to both the west coast and the Chicago area. Heavy fuel oil on the east coast will continue to be obtained from domestic and heavy Venezuelan crudes. Although the product distribution of the gas synthesis plant now under construction is rather inflexible, being approximately 90 pct gasoline (or gasoline plus L.P.G.), the coal hydrogenation process is extremely flexible and can readily produce a high percentage of aviation gasoline, Diesel fuel, jet fuel, or fuel oil, as desired. This factor would favor coal hydrogenation plants for national defense as, in an emergency, aviation fuel requirements undoubtedly would skyrocket. However, some of the gas synthesis primary products can be used in the production of aviation gasoline, and greater flexibility is to be expected for this process in the future as the result of new developments. L.P.G. from the coal hydrogenation process can also be converted to high-test gasoline if desired.

COSTS*

It is estimated that the gas synthesis process using coal can produce 90 pct gasoline and 10 pct heavier products at a cost of 12.4 cents per gallon of total products. Coal hydrogenation products consisting of 16 pct L.P.G., 42 pct gasoline, and 42 pct Diesel oil would cost 12.6 cents per gallon of total products. These figures include all costs, with annual depreciation at 6⅔ pct on the total investment in mines, plants, and pipelines. They do not include a return on the investment, such as interest or profit. Alternatively, coal hydrogenation plants could produce heavy fuel oil for approximately 10 cents per gallon or a high percentage (89 pct) of aviation gasoline and aviation base stock for 15.5 cents per gallon of total liquid products. It is possible that one or the other of these alternatives might be more profitable than the product distribution given above. In these figures, no allowance has been made for byproducts. Separation and sale of chemical byproducts could reduce the net cost of the oil products a few cents per gallon for both the gas synthesis and coal hydrogenation plants. However, as full production of byproducts would flood the market, this credit can be taken only for the first several plants, which may be helped to a considerable extent by the sale of these products. The development of greatly enlarged markets for the chemicals could have a significant effect on the synthetic oil industry as well as on the chemical industry.

Processes

The two principal methods of producing oil from coal are the gas synthesis (Fischer-Tropsch) and hydrogenation (Bergius) processes.

GAS SYNTHESIS

In the gas synthesis or modified Fischer-Tropsch process, the coal first is gasified to produce synthesis gas—a mixture of carbon monoxide and hydrogen. After adjusting the hydrogen to carbon monoxide ratio, the gas is purified and passed over a catalyst

* Cost figures as presented here should not be directly compared with refinery prices of petroleum products, as the latter include return on investment. In the author's opinion an additional 2 cents added to the cost figures given above would provide a liberal margin of profit. If synthetic products were sold at prices equivalent to the cost figures given, the investment would be paid out in 15 years. A margin of 2 cents per gallon would shorten the pay-out time to 10 years.

under suitable conditions of temperature and pressure. The carbon monoxide and hydrogen combine to form chiefly liquid hydrocarbons, with smaller percentages of oxygenated compounds and gases. The products vary over a wide range from solid wax to methane, the relative quantities depending on the catalyst employed, the hydrogen to carbon monoxide ratio, and the conditions of temperature and pressure.

As operated on a commercial scale in Germany with a cobalt catalyst, this process yielded chiefly straight-chain saturated compounds and produced a superior Diesel fuel. The gasoline was not of high quality.

An American improvement in the Fischer-Tropsch process, involving the use of a fluidized catalyst bed and an iron catalyst, is being applied in a commercial plant that now is under construction for converting natural gas to synthetic fuels. This modified process gives a highly olefinic product which can be refined to a high-grade motor fuel. The process does not offer much promise for the direct production of aviation gasoline, but the low-boiling olefins with isomeric low-boiling paraffins from other sources can be used to produce alkylate, an aviation gasoline component.

Byproducts include 5 to 15 pct of oxygenated compounds, mainly alcohols, acids, and ketones, which separate as a water solution. Oil soluble oxygenated compounds also are formed but would normally be converted to hydrocarbons in the refining process.

The operating conditions found suitable for the fluidized process result in a high percentage of gasoline, usually 85 to 90 pct of the total product. The process apparently has little flexibility in this respect. Operations are conducted usually at 600° to 650°F, 250 to 500 psi pressure, with recycle of 3 to 4 volumes of gas per volume of fresh feed, using an iron catalyst.

The Bureau of Mines has carried to the pilot-plant stage an internally cooled converter in which the heat of reaction is removed by flooding a stationary catalyst bed with a cooling oil produced in the process. This process has greater flexibility than the fluidized bed, but the characteristics of the products are similar. It also has the advantage for coal that it can be operated with synthesis gas containing a 1:1 ratio of hydrogen to carbon monoxide and eliminate the oxygen largely as CO_2. With the fluidized bed a high ratio of hydrogen to carbon monoxide, usually 2:1, is used, with elimination of oxygen chiefly as water. Synthesis gas as ordinarily produced from coal would have to be altered by the shift reaction for the fluidized process.

The products expected by these two versions of the gas synthesis process under selected sets of conditions have been estimated as given in Table 1.

Table 1 . . . Products Obtained from Gas Synthesis

	Fluidized Process,[6] Per Cent	Internally Cooled Converter, Per Cent
C_3-C_4	[a]	10
Gasoline	84	50
Diesel fuel	11	20
Cracking stock		10
Wax		10
Fuel oil	5	
Total	100	100

[a] Processed to gasoline.

The Bureau of Mines program includes extensive research and development work on the gas synthesis process at Bruceton, Pa. Investigations are under way on catalysts, converter design, operating variables, and examination of the products. A demonstration plant to produce 80 bbl of finished products per day by the gas synthesis process is being built by the Bureau at Louisiana, Mo. This plant will incorporate converters developed at Bruceton. Fig 2 is a view of a pilot plant at Bruceton employing an internally cooled converter.

Extensive research and development on the gas synthesis process are also being carried on by most of the leading oil companies and by other industrial firms. Industry at present is devoting considerably greater effort to this process than to coal hydrogenation.

Both the Bureau and industry have developed processes that are decided improvements over German commercial practice.

COAL HYDROGENATION

In this process coal is liquefied directly by reacting it with hydrogen under suitable conditions of temperature and pressure in the presence of a catalyst. The coal is first pulverized and mixed with an approximately equal weight of heavy oil from the process and a small percentage of catalyst. The paste so formed is pumped, with hydrogen, through preheaters into a series of high-pressure vessels. Pressure is maintained at 3000 to 10,000 psi and temperatures controlled at 800° to 900°F. Lighter products of the reaction are separated and further hydrogenated in a vapor-phase operation, while the heavy oils are recycled. The mineral matter (usually about 1.1 times ash plus 0.1 sulphur) and nonliquefied portions of the coal are continuously removed from a portion of the recycle oil. Recycling of the heavy oil and vapor-phase hydrogenation of the light oils permit conversion of all the liquefied coal to gasoline and lighter products, although the process also can be operated to produce chiefly heavy fuel oil or Diesel oil. Some methane and ethane are formed, but they are used to provide part of the hydrogen for the process.

The coal hydrogenation process yields a gasoline of high octane number and good rich-mixture performance that meets military aviation-gasoline requirements. It is noteworthy that the Germans made virtually all of their wartime aviation gasoline by hydrogenation of coal and tar, even though 70 pct of their total oil supply was natural petroleum. Coal hydrogenation offers superior gasoline and high process flexibility. Although petroleum processes generally are considered very flexible, there are definite limitations as to the proportions of products. As a result, with the methods now in use, the output of gasoline cannot be increased greatly without an increase in heavier products. With coal hydrogenation, at least 75 pct of the yield may be confined to one product (gasoline, jet fuel, Diesel oil, or fuel oil), and, if desired, gasoline can be produced without any fuel oil production.

Byproducts of coal hydrogenation include industrially valuable tar acids and tar bases. Based on the moisture- and ash-free coal, tar-acid yields vary from 5 to 15 pct and tar bases from 2 to 4 pct. The tar acid increases with decreasing rank of coal. Other portions of the intermediate products could be used for wood preserving and as solvents. If not separated, these byproducts are converted to hydrocarbons in the process and contribute to the yield of liquid fuels.

The Bureau of Mines is conducting research and development work on coal hydrogenation at Bruceton, Pa., where operating variables and catalysts are investigated in autoclaves and pilot plants and radical improvements in the process are being sought. The Bureau's hydrogenation plant at Louisiana, Mo., (Fig 3) is virtually completed. It will

produce 200 bbl a day of gasoline and can be operated to yield other oil products. This plant has a number of improvements on the best German practice. A small number of prominent industrial firms in this country also are conducting research and development on this process, but the work being done by private industry is minor compared to its gas synthesis studies.

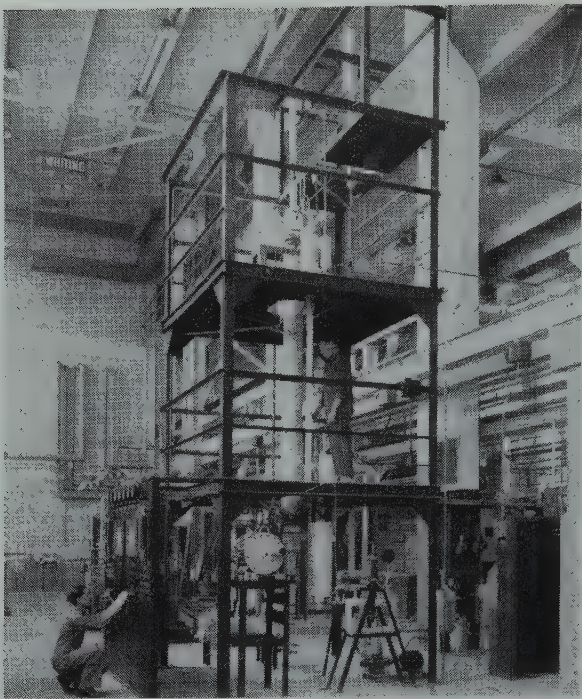

FIG 2—Gas synthesis (Fischer-Tropsch) internally cooled pilot plant. Bureau of Mines synthetic liquid fuels laboratories, Bruceton, Pa.

GASIFICATION OF COAL

In the gas synthesis process, the purified synthesis gas accounts for about 70 pct of the cost of the finished product. In the hydrogenation of coal, the cost of compressed hydrogen represents one-third to one-half the total cost of the product. Coal gasification to produce synthesis gas or hydrogen is thus one of the most important problems in converting coals to synthetic liquid fuels. All synthesis gas produced commercially from coal in this country has been obtained by gasifying byproduct coke in standard water-gas machines. There are several objections to continuing this practice for the production of synthetic liquid fuels:

1. Coking coals in enormous quantities would be required for a large synthetic fuel industry, and such relatively scarce coals should be conserved for the manufacture of metallurgical coke and other preferred uses.

2. Noncoking coals form a large part of the country's coal reserves.

3. A substantial reduction in costs appears to be possible by directly gasifying coal rather than first coking it and then gasifying the coke.

However, in spite of the numerous estimates, such as those given in this paper, on the production of synthetic liquid fuels in this country, the coal gasification processes on which the estimates are based have not yet been carried out commercially with American coals. Furthermore, the direct coal gasification processes used commercially in Germany required special types of fuel that would severely restrict their application in this country.

The Bureau of Mines is playing a leading part in efforts to develop suitable coal-gasification processes for American synthetic fuel plants. At Morgantown, W. Va., the Bureau has laboratories and pilot plants devoted to the gasification of coal and to the purification of synthesis gas. Other Bureau activities in coal gasification include the externally heated retort, the Lurgi pressure-gasification process, and underground gasification. As the gasification of coal in powdered form is expected to be the process more universally applicable, regardless of coal characteristics, the Bureau is making a threefold attack on this problem.

FIG 3—Coal hydrogenation demonstration plant being built by the Bureau of Mines at Louisiana, Mo. Coal is transformed into oil under high pressures and temperatures in the converter stalls (center). Coal preparation unit (upper left) and distillation unit (right).

TYPES OF PROCESSES

The following types of coal-gasification processes are being considered for American synthetic liquid fuel plants:

Gasification of Powdered Coal

At Morgantown, W. Va., the Bureau of Mines has for several months operated a pilot plant[7] gasifying about 20 lb of pulverized coal per hour in oxygen and steam. Although this plant is operated intermittently, its development has reached a stage where it could be operated continuously. The results to date show that gas of synthesis quality can be made by gasifying pulverized bituminous coal.

At Morgantown initial operations have started on a larger pilot plant[8] for gasifying 100 to 1000 lb of pulverized coal per hour in oxygen and steam that has been preheated to the highest possible temperature to minimize the volume of oxygen required. An attempt has been made to design for tempera-

tures as high as 4000°F in the pebble stoves used for preheating steam. Fig 4 is a photograph of the larger Morgantown pilot plant showing, in the foreground, the two Royster pebble stoves for superheating steam.

At Pittsburgh, Pa., the Bureau is investigating the use of the vortex combustor in powdered-coal gasification with steam and oxygen. It has a capacity of 100 lb of coal per hour.

At Louisiana, Mo., a gas synthesis demonstration plant is being constructed for the Bureau of Mines by Koppers Co., Inc. This plant will include a unit for the gasification of powdered coal. The design is based on pilot-plant experiments conducted by the Heinrich Koppers firm in Germany.

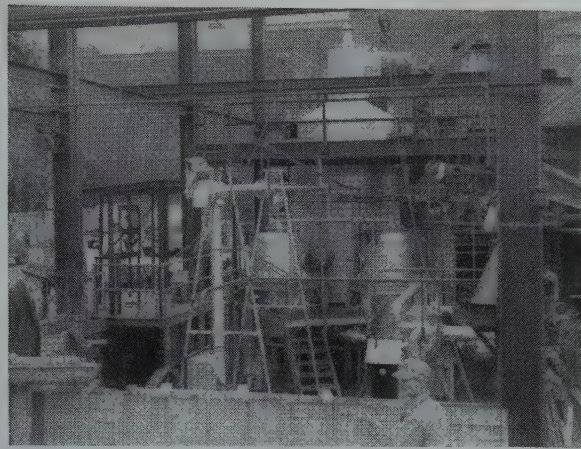

FIG 4—General layout of equipment in the pulverized fuel gasification pilot plant of the Bureau of Mines at Morgantown, W. Va.

FIG 5—Linde-Frankl oxygen plant and 50,000 cu ft gas holder under assembly at Bureau of Mines coal hydrogenation demonstration plant near Louisiana, Mo. Brought from Germany, this 23,000 cu ft-per-hour unit will extract oxygen from the air and is the first of its size to be erected in the United States for coal gasification.

To supply oxygen for this gasification unit the Bureau has a ton-an-hour Linde-Frankl oxygen plant that was brought to this country from Germany (Fig 5).

Gasification in a Fluidized Bed

According to published announcements, the direct gasification of coal in a fluidized bed is being investigated by the Pittsburgh Consolidation Coal Co., working in conjunction with the Standard Oil Development Co. The fluidized coal will be gasified by steam and oxygen.

Externally Heated Retort

At Grand Forks, N. Dak., the Bureau of Mines has a pilot plant in which reactive lignite and subbituminous coal are gasified by steam in an annular, externally heated metal retort (Parry process). This process can produce directly a wide range of gases in varying proportions of hydrogen but requires a sized, reactive fuel and therefore does not have the wide application of the powdered-coal process.

Lurgi Pressure Gasification

At Pittsburgh, the Bureau of Mines is experimenting with the Lurgi pressure-gasification process; however, this process requires a reactive, noncoking, sized fuel, and to obtain this material prior processing is necessary with some coals.

Fixed-bed Process

At Battelle Memorial Institute, coal-gasification experiments are being made with a fixed bed under pressure. It is understood that air is being used to make producer gas for other uses, but the substitution of oxygen and steam for air should give synthesis gas.

Underground Gasification

The Bureau of Mines is actively engaged in both pilot-plant and field-scale experiments on underground gasification of coal. Pilot-plant work is being done at Morgantown, and preliminary work for a second field test has been started at Gorgas, Ala., in conjunction with the Alabama Power Co.

Gasification of Anthracite

The Anthracite Institute laboratories are developing processes for the gasification of fine anthracite to produce synthesis gas.

Low-temperature Carbonization of Coal

Low-temperature carbonization of coal is frequently mentioned as an adjunct to or a substitute for synthetic fuel plants. Low-temperature carbonization is primarily a process to produce a solid, smokeless fuel. Liquid fuels and gas are obtained as byproducts. Low-temperature carbonization of high-volatile coal ordinarily gives the following yields per ton of coal:[9]

Kind of Fuel	Yield
Coke	1,400 to 1,600 lb
Gas (Btu 800–1,000)	3,000 to 5,000 cu ft
Tar	20 to 30 gal
Light oil (for motor fuels)	2.5 to 3.0 gal

The tar produced can be used as a fuel oil or as a raw material for a hydrogenation plant. From the viewpoint of supplying liquid fuels, distillation of the tar beyond the removal of light oil is not very attractive, as 25 to 40 pct is obtained as pitch, while the Diesel oil, produced to the extent of approximately 3 gal per ton of coal, would have a cetane number of only 15 to 20 and would therefore have to be blended with a high-grade Diesel fuel.

There is a possibility of employing

low-temperature carbonization in the large-scale production of synthetic fuels by integrating this process with hydrogenation and Fischer-Tropsch plants. In this case, the tar obtained would go to the hydrogenation plant to be treated along with raw coal, and the low-temperature coke would be used for producing synthesis gas and hydrogen either by the water-gas process or by continuous gasification with oxygen. However, after careful study of various schemes, the Bureau of Mines believes that the direct gasification of powdered coal (without prior low-temperature distillation to obtain the tar) is preferable because of lower investment and operating costs and because of wider application to the different kinds of coal available in this country.

Table 2 . . . Effect of Sulphur Content of Coal on Synthesis Gas Purification Costs

Based on a Plant Producing 10,000 bbl of Oil per Day and 26 lb of Coal per M cu ft of Raw Synthesis Gas

Per cent sulphur in coal	1 pct	3 pct	5 pct
Investment in purification equipment	$3,865,000	$5,447,000	$6,688,000
Recoverable sulphur per day, tons	38	118	198
Total cost of sulphur removal per ton of coal gasified	$0.75	$1.15	$1.52
Net Cost of Purification per Short Ton of Coal Gasified Based on Various Sulphur Credits:[a]			
Credit per long ton sulphur recovered			
$ 5	$0.71	$1.03	$1.32
$10	0.67	0.91	1.12
$15	0.62	0.79	0.91
$20	0.58	0.67	0.71
$25	0.55	0.55	0.51

[a] Sulphur credits include credit for heat recovered from sulphur conversion process, if any.

Characteristics of Coal for Synthetic Fuels

Some characteristics of coals will have a bearing on their use for either the gas synthesis or hydrogenation process. These include ash content, moisture content, storage properties, and, especially, uniformity.

Although high-ash coal can be gasified, it is axiomatic that low ash is advantageous in almost any process using coal and that the value of ash reduction increases with the cost of such processing. Therefore, as gasification for either the gas synthesis or coal hydrogenation process and primary liquefaction in the latter process are expensive, it appears likely that coal washing will be justified.

High surface moisture can cause handling difficulties with fine coal. In any case, the moisture in the coal, if not removed by prior drying, requires heat for evaporation in the processing and therefore may, and very likely will, be a disadvantage.

As synthetic oil plants operate continuously, some coal must ordinarily be stored at the plant to provide for operation when mines or transportation facilities are idle. Coals that weather rapidly will require special facilities for storage.

Uniformity is one characteristic of coal that will doubtless be extremely important for both the gas synthesis and hydrogenation processes. Gasification of coal with oxygen and steam to produce synthesis gas or hydrogen is likely to be a delicately balanced operation that will require a constant feed of uniform-quality fuel. Coal for liquefaction in the hydrogenation process must also be uniform, as variations in ash or petrographic constituents could affect the catalyst requirements and operating conditions.

Cleaning and dewatering, or drying, are thus indicated for synthetic fuel plant coal to ensure reduced ash, controlled moisture, and uniformity. Coal cleaning will also permit mechanical mining, which usually results in excessive impurities in the raw coal.

GAS SYNTHESIS PROCESS

Characteristics of coal that may affect particularly the gas synthesis process are the sulphur content, ash-softening temperature, and coking properties. Rank of coal is not important; any rank from anthracite to lignite can be used.

As the synthesis gas must be virtually free from sulphur compounds, high-sulphur coal will increase the cost of gas purification. Table 2 summarizes recent estimates on the cost of synthesis gas purification. These figures indicate that, where a liberal credit can be obtained for sulphur recovered, high-sulphur coals may be acceptable. If little or no credit is available from the sale of sulphur, reduced purification costs alone might in some instances justify cleaning the coal.

Coal gasification must be carried on at temperatures of at least 1800°F, and it may be desirable to employ considerably higher temperatures to eliminate tar and gum-forming constituents from the gas. With either nonslagging or slagging processes, the ash-fusion temperature is likely to be an important factor. It must be high enough so that the ash will not fuse in the nonslagging process and low enough to fuse completely in the slagging process.

If coal is to be gasified in the fluidized state, as has been proposed, the coking characteristics of the coal may be very important. With strongly coking coal, it may be necessary to recirculate large quantities of noncoking material to prevent agglomeration in the fluidized bed. For the production of synthesis gas by the Parry process, a very reactive noncoking fuel is required, such as lignite or subbituminous coal. Use of the conventional water-gas process would require noncoking fuel, such as anthracite or coke.

Approximately one-eighth of the total coal for a gas synthesis plant is used to produce steam and power. This presents no problem; and, especially in the case of a captive coal mine, it might be a decided advantage as the boiler plant could be designed to handle portions of the coal less desirable for gasification, such as middlings from a cleaning plant, fines that cannot be economically cleaned, or carry-over from the gasification units. If coal is purchased, the most economical steam coal can be used for boiler fuel.

COAL HYDROGENATION PROCESS

The matter of coal selection, or coal distribution in the case of a single source of coal, is much more complicated for a coal hydrogenation plant than for a gas synthesis plant. A commercial plant using the hydrocarbon gases formed in the process as a source of hydrogen would have four different uses for coal, divided approximately as follows:

	Per Cent
Process coal for liquefaction	53
Gasification coal for hydrogen	12
Coal for producer gas	9
Coal for steam and power	26
Total	100

The process coal for liquefaction is, of course, the critical portion. The other uses may in some instances be advantageous, as the equipment may be selected to utilize cheaper coals, or portions of the coal from a captive mine that are less desirable for liquefaction.

Table 3 . . . Fuel and Water Requirements for 10,000 Barrel-per-day Plant

	Hydrogenation	Gas Synthesis
Bituminous coal (11,900 Btu per lb)		
Tons per calendar day	4,300	5,100
Tons per year	1,570,000	1,840,000
For 20-yr life of plant	31,300,000	36,700,000
Daily mine capacity required (at 260 days per year)	6,000	7,100
Subbituminous coal (9,500 Btu per lb)		
Tons per calendar day	5,200	6,400
Tons per year	1,900,000	2,300,000
For 20-yr life of plant	38,000,000	46,000,000
Daily mine capacity required (at 260 days per year)	7,300	9,000
Lignite (7,500 Btu per lb)		
Tons per calendar day	6,600	8,100
Tons per year	2,400,000	2,900,000
For 20-yr life of plant	48,200,000	58,300,000
Daily mine capacity required (at 260 days per year)	9,200	11,300
Anthracite (13,500 Btu per lb)		
Tons per calendar day		4,500
Tons per year		1,619,000
For 20-yr life of plant		32,380,000
Daily mine capacity required (at 260 days per year)		6,300
Water—Minimum stream flow available (spray cooling and recirculation)		
Intake, million gallons per day	9.6	12.1
Net consumption, million gallons per day	6.6	8.1
Water—Unlimited stream flow available (no recirculation)		
Intake, million gallons per day	167.	167.
Net consumption, million gallons per day	0.8	1.

Coal for Liquefaction

Common, banded, bright coals ranging in rank from high-volatile bituminous to lignite are generally suitable for liquefaction by coal hydrogenation. The higher rank medium and low-volatile coals are liquefied to a lesser extent and therefore yield less oil per ton of moisture- and ash-free coal. Anthracite yields virtually no liquid hydrocarbons under hydrogenation conditions.

In addition to the rank of the coal, the petrographic constituents must be considered. Anthraxylon or vitrain in high-volatile and lower-rank coals is almost completely (95 to 98 pct) liquefied. Splint coals give comparatively low liquefaction yields—60 to 87 pct in those tested. Fusain is the least reactive constituent and gives very low yields. Some cannel and boghead coals that are low in fusain and opaque attritus may be suitable.[10]

In general, splint coals should be avoided. Fusain usually can be eliminated to a considerable extent by dedusting or cleaning the coal.

Usually, the lower-rank coals are more readily and more completely liquefied than the higher-rank coals. This tends to offset the disadvantage of high moisture characteristic of the low-rank coals. The yields for various suitable coals in this process are roughly proportional to the heating values of the coal as received. The yields of tar acids are appreciably greater with lower-rank coals.

Mineral matter in coal for hydrogenation may have other disadvantages in addition to its direct detrimental effect in displacing coal in the processing. In the German process, elimination of the mineral matter from the system involved removal of about an equal weight of oil, much of which eventually was lost. Therefore, high mineral matter was more detrimental than would normally be expected. However, processes are being developed which, it is hoped, will result in removal of the mineral matter in a dry material, thus avoiding loss of oil. Sulphur content and ash-fusion temperature are of little consequence in the coal hydrogenation process.

The composition of the mineral matter in coal for hydrogenation may be a factor. Some mineral matter may act as a negative catalyst and require chemicals to neutralize the effect. A high calcium content may contribute to the formation of deposits in the converters. Acidity and chlorine content of the coal affect the quantities and types of chemicals added. Actual tests are usually needed to measure the extent of these influences, which usually are not serious.

Coal for Gasification

The requirements for gasification coal to produce hydrogen are the same as for the production of synthesis gas, except that sulphur content is of less importance, as gas purification is optional and, when used, would not have to be carried to the low concentration of sulphur compounds required for the gas synthesis process.

Coal for Producer Gas

Fuel gas for heating purposes in the plant can be made in conventional air-blown gas producers if suitable fuel is available. Gas producers operate best on a sized fuel with fairly high ash-softening temperature. Low-rank, noncoking coals that decrepitate to fines on heating might not be suitable fuels. If no suitable fuel is available, special gasification equipment might be required, or one of the synthesis gas generators might be operated with air instead of oxygen to produce cheap fuel gas.

Coal for Steam and Power

Boiler house equipment would be selected, as in the gas synthesis process, to burn coal less desirable for other uses—middlings from the cleaning plant or fines—or, in the case of purchased coals, the most economical steam coal available.

Nation-wide Survey for Synthetic Fuel Plant Locations

At the request of the Secretary of the Interior, the Corps of Engineers of the Department of the Army is making a survey for the Bureau of Mines to determine suitable areas in the United States and Alaska for the location of synthetic liquid fuels plants. A contract already has been let for the first part of this work covering the following areas: northwestern Colorado, western Kentucky, and a part of southeastern Texas.

Economic and Geographical Factors

Some of the factors to be considered in attempting to predict which coals will be preferred for large-scale production of synthetic liquid fuels include:

1. A plant producing some 10,000 bbl of oil per day will require the quantities of coal and water shown in Table 3, both of which must be assured for the entire life of the plant. For larger plants the requirements would be roughly proportional to the output. Numerous locations in the United States have ample coal reserves for commercial synthetic fuel plants, but in the western states that have the largest coal reserves, water supplies may restrict the number and size of plants.

2. As a wide range of coals can be used, preference doubtless would be

given those with a low cost per million Btu delivered to the plant. This would tend to eliminate coals that command higher prices, such as the metallurgical coking coals, the prepared sizes of anthracite, and coals with high mining costs. It would also favor coals close to suitable plant sites.

3. Where new mines must be opened, strip mining offers such advantages over most deep mines as fewer men to train and house and more rapid development to full production.

4. For the gas synthesis process, lower-rank coals and lignites at the same cost per million Btu would be less desirable than higher-rank coals. Because moisture increases the cost of handling and drying, low-rank, high-moisture fuels must be made available at a lower cost per million Btu. However, in the case of coal hydrogenation, compensating factors partly, if not completely, offset the disadvantages of low-rank coals. These coals are more reactive and generally liquefy more completely than high-rank coals. Consequently, conversion costs are lowered enough to offset increased handling and drying costs. This factor improves the competitive position of hydrogenation for the lower-rank fuels.

5. Proximity to markets and transportation facilities undoubtedly will be factors.

6. Availability of suitable labor would be important. While housing might have to be provided in almost any geographical area, regions remote from centers of population would require the importation of workers and the establishment of complete towns with churches and schools in addition to houses.

7. Variations in climate through the coal fields of the United States are not great enough to affect plant operations seriously. Alaska, on the other hand, doubtless would present some difficulties due to the long, extremely cold winters.

8. The capital investment for plants in the western states, such as Wyoming, North Dakota, Colorado, and Montana, which have the greatest coal reserves in the United States, would be higher than for plants in some eastern locations. The distance from steel and equipment manufacturing centers and the need for long pipelines to the major markets increase construction costs. The occurrence in the west of thick coal seams that can be strip mined at low cost with reduced manpower is a compensating factor that might more than offset the higher initial plant and pipeline cost.

9. National defense considerations might oppose the concentration of great plant capacity in one area or erection of a plant close to an important target.

10. The first plants constructed are likely to be integrated with other operations such as the chemical or gas industries. This would favor the use of coals that are reasonably close to large chemical and natural gas markets.

11. The existence of available facilities, such as developed mining capacity or a coke plant of sufficient size to provide coke and coke-oven gas for the production of hydrogen or synthesis gas, might readily influence the location of the initial plants.

Concluding Statement

More important than the question as to which coal will be used for the initial synthetic fuel plants is the question of when they will be built. The need for prompt erection of at least a few commercial plants was aptly brought out by R. A. Cattell, chief of the Petroleum and Natural Gas Branch, Bureau of Mines, in a recent letter,[11] part of which is quoted here:

> Synthetic liquid fuels are not going to do us very much good in an emergency if we have to start from scratch, or from an unduly small nucleus. I often think of our experience in helium production. In World War 1 helium production was started from scratch, and only 200,000 cubic feet had been produced when the Armistice was declared. When World War II started, we had one plant and the output was small, but we had a well-trained staff and crew of men, and the wheels were turning. We were able to expand from that one plant to five plants, and increase the production 22 fold. The needs for helium were met as fast as they developed. During World War II our production in 12 hours exceeded the 200,000 cubic feet that was produced during all of World War I.

Acknowledgments

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Drilling and Sampling Unconsolidated Materials

By LEON W. DUPUY,* Member AIME

Introduction

Many articles have been written describing peculiar and particular types of drilling. Little correlation has been made between the character of ground to be drilled and sampled and the type of drilling most desirable for procuring satisfactory samples. In development work, the engineer and metallurgist often desire the recovery of a sample in as physically natural a condition as possible, particularly when the sample must provide material for process tests as well as chemical and physical analyses. This paper describes the results of experimentation in drilling unconsolidated materials carrying titanium minerals in the Magnet Cove area, Ark. The drilling was most difficult in that the ground varied much in consistency, and it was vitally necessary to recover a physically natural sample. Ordinary churn drilling recovered a sample that could be analyzed chemically, but the sample was not in its natural state. The problem was successfully solved by utilizing and further developing methods used in the oil and clay industries.

In the Magnet Cove area of Hot Spring County, Ark., the titanium minerals include rutile, brookite, anatase, leucoxene, and others. They occur in soil, clays, and altered rocks adjacent to novaculite in contact metamorphic deposits. In preparing titanium minerals for use in the industry, it is necessary to know the physical characteristics of the ore. Also it is believed that the particular physical character of the titanium mineral used is just as important as the chemical character. Brookite and rutile, for example, are the same chemically but differ in crystalline structure. In preparing titanium minerals for market, beneficiation is necessary. For the millman to make a satisfactory laboratory test, the sample he uses must be as nearly of the same physical character as he would have from mine run ore.

In the past at Magnet Cove, churn drilling had been done on the properties, and assays of total titania had been obtained. It was found, however, that the total titania was indicative merely of the presence of titania but did not tell the metallurgist the character of the material or even give him the slightest indication of what he might expect to recover. When the Bureau of Mines entered the field in the winter of 1947 and 1948, it was therefore necessary that a natural sample be recovered.

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LIMITATIONS IN ORDINARY PRACTICE

The churn drilling that had previously been done had shown that the occurrence of titanium-bearing mineral was somewhat widespread. The samples were, however, completely beaten up and in a finely powdered state. The minerals had been slimed, and such ore dressing tests as were made from the churn-drill samples were found to be worthless.

Test pitting and outcrop sampling had provided satisfactory samples of surface ore but did not represent the entire ore body. A quantity of rutile had been produced from the surface residual concentrations in the westerly part of the cove, but production ran into metallurgical trouble as depth was attained. Representative deep samples upon which ore dressing research might depend were needed.

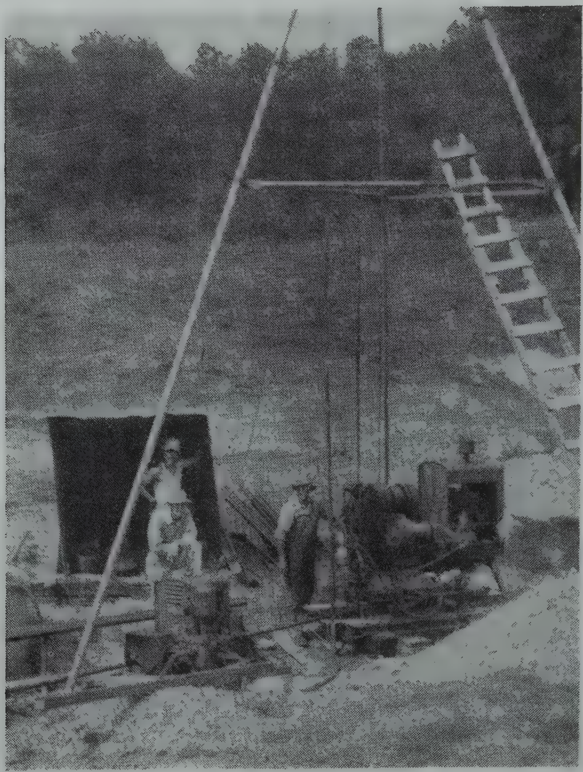

FIG 1—Diamond drill and water pump used at Magnet Cove. Set-up using portable iron pipe tripod.

Diamond Drill Experience and Experimentation

First it was decided that diamond drilling might be the best method of approaching the problem. In the eastern part of the cove the ore near the surface was a residual material, including soil, clay, and quartz boulders containing very jagged crystalline edges. A shot drill was, of course, out of the question because of the washing effect of the drilling water and the fact that the core would therefore be completely ground up. Using a diamond drill with a normal core barrel, it was found that the water washing between the end of the inner tube and the bit completely destroyed the core. It was decided to use the newer type of core barrel with a long inner tube similar to that manufactured by several firms.

DRILL USED

The diamond drill used by the Bureau of Mines at Magnet Cove was a Longyear Junior Strateline equipped with standard tools. Subsequently, a cathead for drive piping was installed on the machine. Fig 1 pictures the equipment utilized.

FIG 2—Diamond drill equipped to use cathead for drive piping.

CASING BIT

At one time during the operation the driller believed that he could attain good core recovery by using a casing bit and just pushing it down through the soil and rotating it a little when he hit a boulder. The hard quartz boulders quickly destroyed the casing bits, and core recovery was nonexistent.

CORE BARREL

The core barrel adopted for the project was the Series M barrel equipped with a long inner tube. Use of the barrels on the unconsolidated ground showed first that the rotation speed and pressure on the bit have a great deal to do with recovery. After much experimentation, it was found that the most generally satisfactory method of drilling in soft ground was to "feel it out" and to use reasonably slow rotation and a minimum volume of water. High rotation speed was not practical, since the quartz boulders lying in the unconsolidated material were rough, jagged, and hard. When a rapidly rotating bit contacted such a rough, hard object, the diamonds were torn out almost immediately. There were instances where the bit was completely ruined in a few revolutions. High speed also requires much more water. High water pressure—that is, using abundant water—washed away the core. Even with the long inner tube the washing of the solutions in the bottom of the hole was too vigorous to permit the slowly advancing bit to capture the soft material before washing it away. In these instances, of course, sludges were saved and comparisons made between the character of the core and the character of the sludge. Holes were started at NX size and reduced to BX when necessary. It is possible that using sizes larger than NX might have resulted in better recovery of core. The high cost of larger barrels, bits, and accessories than NX was the principal reason for starting NX. The losses of material were principally in the loose, unconsolidated soil and clay. A number of holes were drilled following the preferred practice described, but the operation could hardly be called successful. Utilizing the slow rotation and minimum water method, core recovery occasionally amounted to 100 pct but on the whole was little better than 50 or 60 pct most of the time.

USE OF DRIVE PIPE WITH ROTARY CORE DRILLING

A page was borrowed from the book of clay sampling. A cathead was rigged on the side of the drill (Fig 2), and the drive-pipe method of sampling utilized during the penetration of the soft material. The first runs, when no cathead

FIG 3—Bucket drill moving onto new set-up.

was available, were made by taking the wire line off the hoist drum and replacing it with a manila rope. Drive piping was then done, using the hoist drum as a cathead. The drive pipe was run until it was well-seated in a hard boulder before changing to rotary diamond drilling. The cutting shoe of the drive pipe made a seat in the boulder or rock from which the diamond drill could take off without running on any jagged edges. BX casing was used for the drive pipe. The shoe was the same size as the BX casing, so that when a boulder or hard rock was encountered and the drive pipe pulled, the rock could then be drilled with a standard NX bit. The speed of drilling was increased materially, and it was also possible to use heavier pressure when desired. As soon as the hard rock was passed, the drive pipe was put back in the hole, and the hole was sunk drive-pipe fashion down to the next hard stratum or boulder, then diamond-cored through the boulder, and so on, to the end of the hole. With the adaptation of the combination drive pipe and rotary method, it has usually been possible to recover 100 pct of the core in natural and true form and to disregard sludges. When diamond drilling, sludges were still saved in the settling tubs as a matter of insurance in case something went wrong and until after the sample was inspected. If the core sample was satisfactory, the sludge was thrown away. In some instances, portions of the core have been lost by washing when small, soft clay deposits exist inside the harder boulders or between layers of harder rock. Usually, however, the driller was able to make a fair job of core recovery. The one difficulty with the method is that the average diamond driller dislikes drive piping very much and will use every excuse conceivable to avoid it, for it is hard work. With the right kind of a man on the rig, one who can be persuaded that core recovery is what is desired, core recovery will be procured.

Using the combination drive pipe and diamond drill, a driller and helper are required. A sampler can take care of two rigs, so that labor requirements per rig are $2\frac{1}{2}$ man-shifts. Using the drill crew as samplers reduces footage 20 to 30 pct. The rig averages about 13 ft per shift, including moves and drilling holes 170 ft deep. Runs are limited to 3 ft, except in consolidated material.

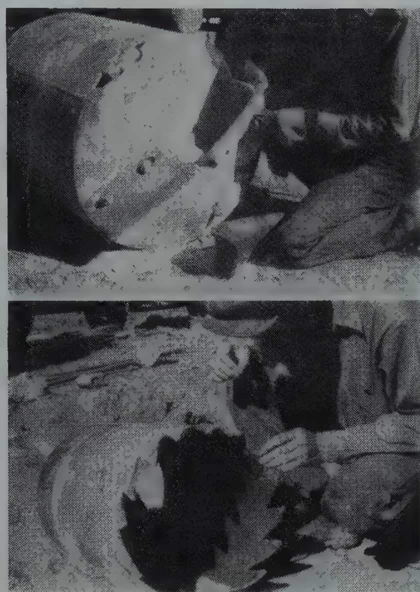

FIG 4—Typical buckets. *Top* Door-type bucket. *Bottom* Sawtooth bucket.

Before use of the drive pipe with runs limited to 2-ft lengths, the diamond drill averaged 7 ft per shift.

While experiments with a diamond drill were being conducted, a second method of coring, that of utilizing a Baker cable-tool core barrel with a bucket drill and a churn drill, was successfully developed.

Bucket-drill Experience and Experiments Adapting the Baker Core Tool

THE BUCKET DRILL

The bucket drill has been described in several Bureau of Mines publications.[1,2,3] Former Bureau projects, for example, at Coso Hot Springs[1] utilized a bucket drill with a simple rotation utilizing ring gear, cross head, and Kelly bar. The rig used at Magnet Cove was a variation of that earlier machine. It includes not only the features of the original bucket drill with the ring-gear drive but also had an auxiliary churn-drill device that could be utilized to break boulders and do some churn drilling. The rig was mounted on a four-wheel-drive truck and was quite mobile (Fig 3). Fig 4 shows some of the tools utilized in sinking the holes.

USE OF TOOL

The equipment gets its name (bucket drill) from the bucket-type appearance of the main tools, such as the trap-door-bottom bucket shown in Fig 4 *top*. The buckets may be of any diameter desired. In this particular instance the diameter was 20 in. The trap-door bottom acts as a sort of auger bit which peels the clay and small boulders out of the hole up into the bucket. Usually, about 12 in. is cut on each trip of the tools into the hole. Another useful tool was the sawtooth bucket, Fig 4 *bottom*. The sawtooth bucket acts somewhat as a crosscut saw. It cuts a solid core through reasonably consolidated material, that is, material that ordinarily is considered soft or material that might be mined with a power shovel, perhaps without blasting, but yet is not too hard. There have been times when real hard rock, such as flint, has been cut with sawtooth buckets though cutting very hard material is not economical.

FIG 5—Swinging the mother hubbard bit.

[1] References are at the end of the paper.

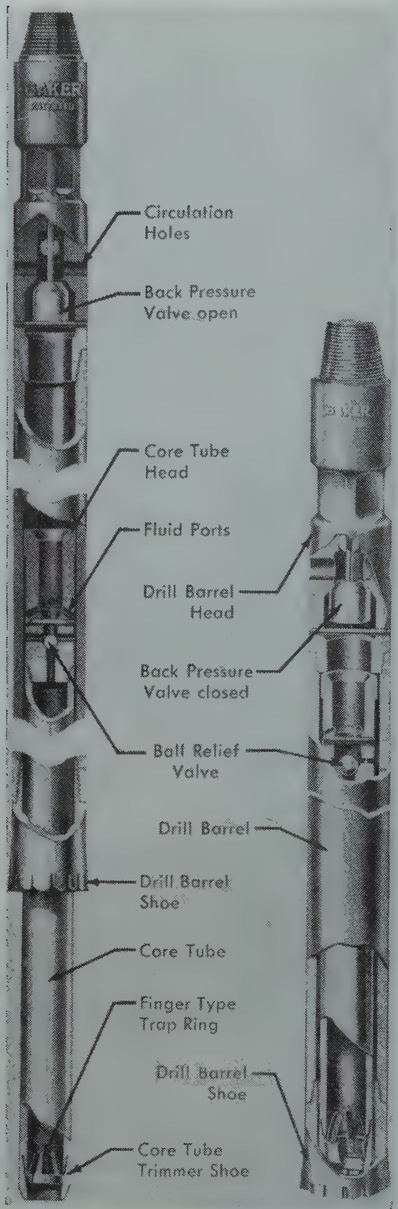

FIG 6—Baker cable tool core barrel.

The sawtooth bucket is probably used more than any other on bucket-drilling jobs because it is simpler to manipulate and will handle boulders much better than the trap-door bucket. In some instances, if water is in the hole, the trap-door bucket is equipped with flaps of flexible material on the inside, so that the pressure of the water as the bucket is pulled out of the hole inside the bucket will shut the flaps and make it possible to withdraw a bucketful of mud. Otherwise, the mud will run out the bottom of the bucket. A number of other tools are also used, but the two mentioned are perhaps the most common buckets. A rock bucket used by a California firm consists of heavy straight steel teeth somewhat like shovel teeth inserted on the side of the bucket with the flat edge parallel to the circumference and the long part of the tooth flush with the outside of the bucket. It has been quite successful in drilling through cobblestones and cemented material in foundation work.

The greatest bugaboos in bucket drilling are handling or passing through the occasional large, hard boulders. In the Magnet Cove work a "mother hubbard bit," shown in Fig 5, was acquired from the oil fields and adapted to the bucket rig. The mother hubbard bit in this instance weighed about 1400 lb, and the bit face was dressed to 20 in. It was utilized with the churn-drill mechanism with which the rig was equipped. The heavy bit was a little hard to hold with the comparatively small brake drums on the bucket drill but was very effective in shattering such large boulders or hard ribs as were found. The unsatisfactory thing about the use of a mother hubbard bit was that it was getting back to churn drilling, and the physical character of the sample was being changed, which was a feature not wanted.

BAKER CABLE-TOOL CORE BARREL

The Baker cable-tool core barrel, details of which are shown in Fig 6 and 7, is a tool that has been used in the oil fields of Pennsylvania and other parts of the country,[4,5] and sometimes is called a wire-line core barrel, although the latter name is used for other designs of core barrels also. In ordinary mine-sampling work the Baker tool is relatively unknown. The tool was tried when it was found that it was necessary to use a mother hubbard bit in the bucket-drill holes too frequently.

The core barrel (Fig 6) consists of two main parts—an outer drilling barrel and an inner core-retaining tube, or inner barrel. The inner tube does not move upward at any time after the coring is begun. It rests on the bottom of the hole until the tool is pulled out of the hole. An outer drilling barrel slides up and down on the inner tube and cuts away the formation around it by churn-drill action, permitting the inner tube to drop. As shown in Fig 6, the top of the inner tube is flared a little, permitting the bottom of the drill-barrel head to tap it should it hang up a little in sliding down over the core. Thus, in effect, the inner tube is a drive pipe.

An important feature of the inner tube is the ball-check valve, or a relief valve, that permits any water trapped in the core tube or inner barrel to escape as the tube fills with core. The ball valve also prevents any fluid under heavy pressure from above from acting upon the core.

The parts that make up the assembly are illustrated and labeled in Fig 6 and 7. The drill-barrel shoe is made with a clearance in the cutting edge which allows the barrel to drop freely in the hole. The cutting teeth of the drill shoe are staggered to prevent key seating and are faced with hard metal.

The drill shoe or bit is connected to the lower end of the drill barrel with a special tool joint thread. Since the inside diameter of the shoe is smaller than the inside diameter of the drill barrel, the shoulder thus formed prevents un-

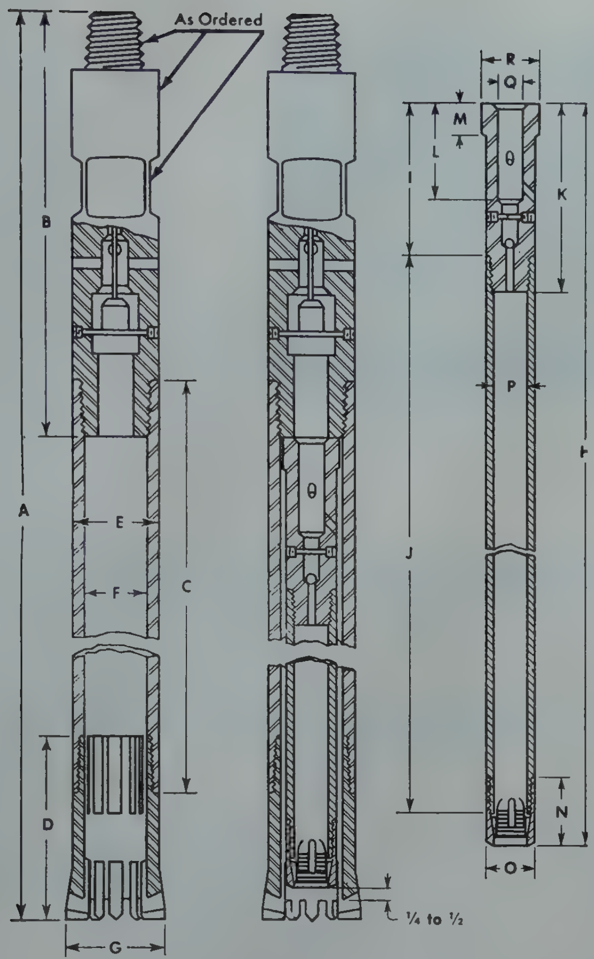

FIG 7—Baker cable tool core barrel.

Dimensional Data for Baker Cable Tool Core Barrel.

Size of Barrel	A	B	C	D	E	F	G	H	I
5	11 ft. 8 in.	32	8 ft. 6 in.	12	4⅜	3¼	Gauge Ordered	8 ft. 10½ in.	13½
6	11 ft. 8 in.	32	8 ft. 6 in.	12	5¼	4		8 ft. 10½ in.	13½
Size of Barrel	**J**	**K**	**L**	**M**	**N**	**O**	**P**	**Q**	**R**
5	7 ft. 7 in.	15	8	2½	3½	2¾	2⅛	2	3⅛
6	7 ft. 7 in.	15	8	2½	3½	3 7/16	2 13/16	2	3⅞

intentional removal of the core-retaining tube (inner barrel) which is provided with a shoulder at its upper end. The inner surface of the drilling shoe has watercourse grooves to allow free passage of fluid from within the drill barrel to the cutting edge of the drill shoe. The core retaining spring, or ring, fits inside the inner tube and prevents the dropping of core.

The opening of the valve in the drilling barrel (outer barrel) at the upstroke permits fluid to enter the barrel, and the closing of the valve in the core retaining tube on the downstroke prevents the fluid from entering the core retaining tube. This action on the downstroke which closes both valves produces a hydraulic effect which directs the fluid between the inner barrel and the outer barrel with such force that it keeps the teeth of the drill shoe free from cuttings, and often makes it possible, particularly when a new drill barrel shoe is used, to drill certain formations as fast or faster than with a standard churn-drill bit. It will be noticed in the sketches that ports in the barrel permit the free entrance of the drilling solutions or muds when the outer barrel valve is open.

The best results were obtained by using an intermediate churn-drill stroke (3 to 4 ft) rather than a long or a short stroke, and a drilling speed of 32 to 35 strokes per minute. Although the manufacturer recommends using a stiff rope socket, the best results were obtained with a swivel socket, unblocked, permitting the outer barrel to rotate.

In actual practice a few coarse cuttings usually settle on top of the solid core within the core barrel. These coarse cuttings are useful for panning in the field to determine if the material being cut carries mineralization. In going through clays and sticky material, the core often wedges tight within the barrel, and a hydraulic hand pump is used to extract the core in the barrel.

FIG 8—Attaching hydraulic pump to core barrel.

A picture of the hand pump being connected to the core barrel preparatory to extracting the core is shown in Fig 8. When the core is extracted, it is usually found that a certain amount of sludge adheres to the core, and it is often necessary to scrub the core clean with a wire brush to keep from contaminating the sample. At other times the material is scraped off with a knife. With hard-rock cores simple washing is sufficient. After the core is extracted, the procedure of handling, of course, is familiar to all.

When drilling through soft clay, it is often found that the true drive-pipe action has taken place and that the core has somewhat elongated in the barrel. It is easy enough to compensate for that elongation by calculation should it become necessary to divide the core at any particular interval.

At the Magnet Cove project wear and tear on the core barrel have been comparatively light, although the spare parts are always carried in stock as a guarantee against shut-downs. The cost of operations utilizing the core barrel depends largely upon the efficiency of the drill crew, the character of the rig, the weight of tools, the depth of hole, and the character of the ground. The use of the hydraulic hand pump for removing core is a little tedious and time-consuming. Much time is saved if a second inner tube is ready and assembled so that the driller can proceed with the next core run while the sampler and drill helper remove the core from the last run. The hole is always bailed as soon as the run is finished to keep heavy particles from settling out and interfering with the next run. Changing the bit and outer barrel to the waiting empty inner tube requires 5 to 10 min and is enough time to permit the settling of material that would cause trouble.

To use the cable-tool core barrel for depths over 100 ft in soft materials, graduated sizes of barrels and bits should be employed.

A normal churn-drill crew of 2 men, driller and helper, is required for rig operation. One sampler can normally take care of 2 rigs. Hence, one drill shift requires 2½ man-shifts plus the normal engineering supervision. Drilling speed has varied appreciably. In a shallow hole (under 50 ft) the driller often makes 20 to 25 ft per 8-hr shift. In deeper work 10 to 15 ft of drilling in a shift is more common. For October, with 44 drill shifts, a standard-type churn-drill rig, using a core barrel, drilled 517 ft in 5 holes, including moving time. As experience is obtained, performance is becoming better.

USE OF STANDARD TYPE CHURN DRILL

Since the bucket drill's churn-drill mechanism proved the value of the cable-tool core barrel but was not designed for continuous churn drilling, a regular standard churn drill was moved in. Operations with a standard churn-drill rig have been found to be far more satisfactory than with the bucket-drill rig. The speed of drilling was almost doubled.

Conclusions

The bucket drill, when doing the work for which it was designed, recovers a large sample with its auger-type bucket and with the sawtooth bucket. That sample, of course, is very valuable, particularly for metallurgical work. To have a large sample and one that will supply ample bulk for the mill testing laboratory is a very desirable result. The cable-tool core barrel has proved quite satisfactory in providing core in unconsolidated material starting from the surface. It is more satisfactory for development purposes than the bucket drill in ascertaining the extent of an ore body of unconsolidated material where conditions are relatively unknown and where transition from hard to soft material is rapid. Where a comparatively large sample is desired, the 3 in. diam sample provides a relatively large core which can be properly split and provide a satisfactory metallurgical sample. The tool is a valuable adjunct of the standard churn drill and in development work could well be profitably utilized more frequently.

The diamond drill, with its drive-pipe combination in unconsolidated ground, gave équally as good results as the cable-tool core barrel but required more physical exertion and was less foolproof.

Acknowledgments

Grateful acknowledgment is given to A. B. Needham, Donald F. Reed, S. P. Holt, and George W. Lester, mining engineers of the Bureau of Mines, for active assistance and participation on the project; to J. R. Thoenen, chief of the Tuscaloosa Branch, for his assistance and consultation; and to Lowell B. Moon, chief of the Mining Division of the Bureau of Mines, for his aid and counsel in the prosecution of the projects.

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Effects of Rod Mill Speed at Tennessee Copper Company

By J. F. MYERS* and F. M. LEWIS,* Members AIME

The purpose of the mill tests reported herein, was to determine the relative power efficiency of fast and slow rod mill speeds on the ores of the Tennessee Copper Co.

The tests were carried out at the Isabella Mill of the Tennessee Copper Co. under normal operating conditions. Every effort was made to keep the ore supply and other conditions constant, but it will be noted that some slight variation did occur.

The open circuit 6 by 9 rod mill, and ore, used in these tests have been previously described.[1] Four motor drive pinions were available and the tests were performed in the following manner: each pinion was used to drive the mill for a period of one week. Constant sampling prevailed during the test run, and a composite sample for each speed test was made. Realizing that some slight liner wear took place during the four-week test-run, and that some slight variation of the ore may have taken place, a second and then a third series of four-week test-runs were made with the four different pinions.

The screen analysis of each of the three-week run samples at the various speeds were composited and the results are shown in Table 1.

The authors feel that the 12 weeks of operation consumed during these test-runs would represent fairly the effect of each mill speed. The data of line 37 show that within the error of sampling and plant operation, the tons of minus 65-mesh material generated per horsepower hour was the same at all four speeds.

Obviously, rod and liner consumption could not be determined for each mill speed in so short a time. The rod load was maintained at a uniform level throughout the series of tests.

The authors have operated rod mills at various speeds over the past several years, and it is their observation that rod consumption is directly proportional to the mill speed. Likewise, that the rod consumption per unit of work performed is a constant, providing that all other operating factors remain the same.

It is a well known fact that the efficiency increases in a ball mill as the speed decreases. This is due to lateral segregation of the large and

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[1] References are at the end of the paper.

Table 1 . . . Data on Various Rod Mill Speeds Isabella 6 by 9 Rod Mill—2½ In. Rods

Line Number		A	B	C	D
1	Tons per day	1,100	1,100	1,100	1,100
2	RPM	26.5	24.0	22.7	21.4
3	Discharge opening, in.	24	24	24	24
4	Horsepower input	194	181	175	170
5	Mill dilution	78.9	79.0	78.8	78.6
6	Critical speed, pct	82.4	74.6	70.5	66.5
	Mill Feed: Mesh	Weight, Pct	Weight, Pct	Weight, Pct	Weight, Pct
7	0.742	3.3	5.7	2.9	3.1
8	0.525	21.2	15.6	13.5	10.4
9	0.371	18.3	16.1	18.9	18.4
10	3	13.5	12.1	13.6	13.0
11	4	8.7	7.9	8.5	9.1
12	6	6.0	6.0	5.8	6.7
13	8	3.7	4.2	4.7	5.1
14	10	3.0	3.7	3.7	4.1
15	14	2.2	2.9	2.8	3.2
16	20	2.0	2.8	3.0	3.1
17	28	2.0	2.9	2.8	3.1
18	35	1.9	2.9	2.6	2.9
19	48	1.9	2.9	2.5	2.8
20	65	2.0	2.7	2.6	2.9
21	100	1.8	2.1	2.2	2.4
22	200	2.4	3.0	3.3	3.4
23	−200	6.1	6.5	6.6	6.3
	Mill Discharge: Mesh				
24	14	1.8	1.6	1.4	1.8
25	20	4.0	3.2	3.7	4.4
26	28	7.5	6.6	7.0	7.7
27	35	10.6	11.1	10.7	11.0
28	48	11.2	12.2	11.9	11.5
29	65	11.0	12.5	13.2	12.9
30	100	11.3	10.9	10.5	10.1
31	200	13.6	14.2	15.0	14.2
32	−200	29.0	27.7	26.6	26.4
	Tons, 65 Mesh Material				
33	In feed	113.3	127.6	133.1	133.1
34	In product	592.9	580.8	573.1	557.7
35	Generated	479.6	453.2	440.0	424.6
36	Generator per hour	19.98	18.88	18.33	17.69
37	Per hp-hr	0.1030	0.1043	0.1047	0.1041

small balls.[2] Balls respond individually to mill speeds and other factors. Rods, being rigid from end to end, show no tendency to segregate according to size, and hence, respond directly to speed and/or increased contact with each other and the mill shell.[1]

Conclusions

The conclusion reached (see line 37 of Table 1) is that with all other factors the same, the work accomplished in a fine crushing rod mill is directly proportional to the rotating speed.

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Temperature Compensation of Old Type Askania Magnetometers

By T. KOULOMZINE,* Member AIME

Introduction

The theory of the Askania magnetometer, as well as a complete discussion of all factors influencing magnetometer readings, is very ably described by J. Wallace Joyce.[1] We will assume that the reader is thoroughly familiar with this theory and we will use whenever possible the same notations for the physical values involved in the theory. On the other hand, we believe that the principal formulas of the magnetometer can be deduced much more easily by the use of differential calculus than by the methods employed by Joyce. Consequently, we are presenting herewith a resumé of the theory of the vertical magnetometer in order to determine the correct formula of the temperature coefficient which we propose to compensate.

Short Theory of the Vertical Magnetometers

We shall accept the following notations:

m = total mass of the magnet system, in grams.

g = acceleration of gravity, in dynes.

a = projection on the axis of the magnet system of the distance between the axis of support and the center of gravity of the magnet system, in centimeters.

h = projection on the perpendicular to the axis of the magnet system of the distance between the axis of support and the center of gravity of the magnet system, in centimeters.

$M = 2pL = M_o$ magnetic moment of the magnet system.

Z = vertical component of the earth's magnetic field.

θ = deflection angle of the magnet system.

= focal length of objective lens in the optical system

S = scale reading.

In a vertical magnetometer the axis of the magnet system is kept horizontal within 2°. Therefore, referring to Fig 1, we see that the magnetic torque of the magnet system influenced by Z must be counterbalanced by the gravity torque.

$$MZ \cos \theta = mga \cos \theta + mgh \sin \theta \quad [1]$$

or dividing by $\cos \theta$

$$MZ = mga + mgh \tan \theta \quad [2]$$

If we now move to another magnetic station and the temperature conditions change, the values of all the variables will change also:

M, a, and h under the influence of temperature.

Z, both with time and space.

θ and S, as a result of all the above changes.

Taking increments in Eq 2 we obtain:

$$(M_o + \Delta M)(Z + \Delta Z) = mga_o + mg\Delta a + mg(h + \Delta h) \cdot \tan (\theta + \Delta\theta) \quad [3]$$

but $\tan (\theta + \Delta\theta)$, because θ is small, can be replaced by $\tan \theta + \tan \Delta\theta$; in fact

$$\tan (\theta + \Delta\theta) = \frac{\sin \theta \cdot \cos \Delta\theta + \cos \theta \cdot \sin \Delta\theta}{\cos \theta \cdot \cos \Delta\theta - \sin \theta \cdot \sin \Delta\theta}$$

but $\sin \theta \sin \Delta\theta \to 0$

$\tan (\theta + \Delta\theta) \to \tan \theta + \tan \Delta\theta$. Furthermore, neglecting all terms of the equation containing two differentials or a differential and a $\tan \Delta\theta$, Eq 3 becomes:

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[1] References are at the end of the paper.

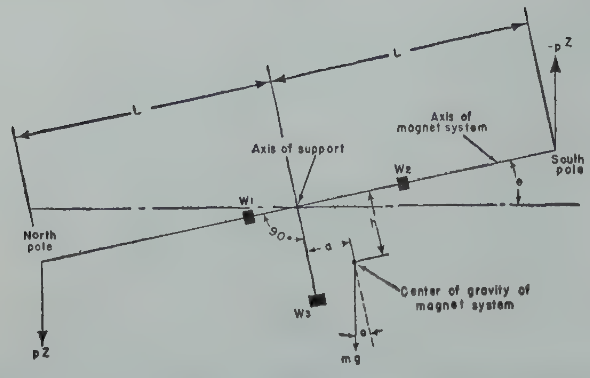

FIG 1—A vertical magnetometer. In this drawing distances a and h are very much exaggerated. In a real magnetometer they are of the order of 0.01 cm.

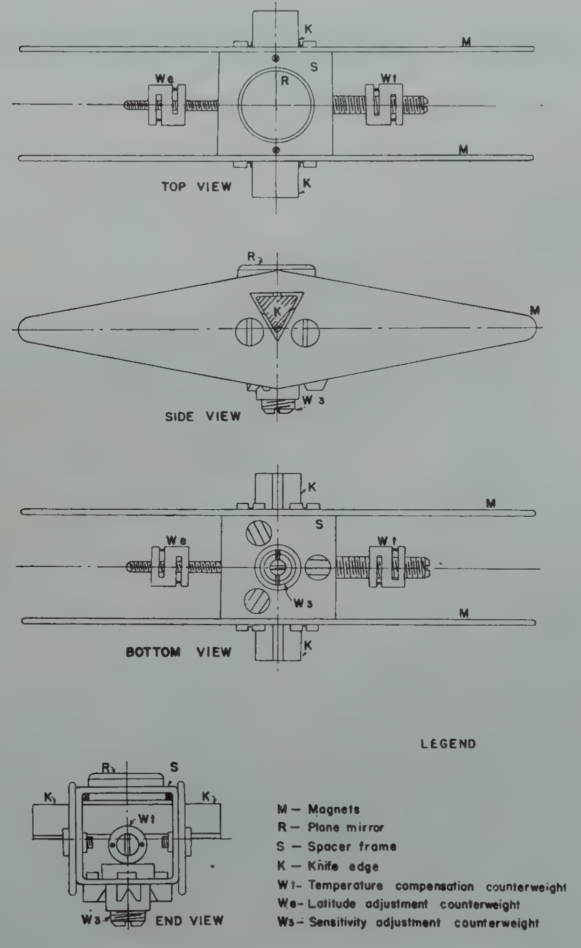

FIG 2—Diagram of modern temperature-compensated magnetic system.

$$M_oZ + Z\Delta M + M_o\Delta Z = mga_o + mg\Delta a + mgh \tan \theta + mgh \tan \Delta\theta \quad [4]$$

subtracting Eq 2 from Eq 4 we obtain:

$$Z\Delta M + M_o\Delta Z = mg\Delta a + mgh \tan\Delta\theta \quad [5]$$

or

$$\Delta Z = \frac{mgh}{M_o} \tan \Delta\theta + mg \frac{\Delta a}{M_o} - Z \frac{\Delta M}{M_o} \quad [6]$$

This is the basic equation of the magnetometer.

It can be shown easily from the study of the optical system that $\tan \Delta\theta = \frac{S - S_o}{2f}$, while Δa and ΔM vary only with temperature changes; therefore

$$\Delta Z = \epsilon_z(S - S_o) + T_c(t - t_o) \quad [7]$$

where

$$\epsilon_z = \frac{mgh}{2fM_o} \quad [8]$$

is the scale coefficient of the magnetometer

and

$$T_c = -Zu + \frac{mga_o}{M_o} p \quad [9]$$

is the temperature coefficient of the instrument

$u = \frac{\Delta M}{M_o} \frac{1}{t - t_o}$ being the temperature coefficient of the magnetic moment

and

$p = \frac{\Delta a}{a_o} \frac{1}{t - t_o}$ The temperature coefficient of the distance between the center of rotation and the projection of the position of the center of gravity on the axis of the magnet system.

Furthermore from Eq 2 we find that when θ is small or zero,

$$a_o = \frac{M_oZ}{mg} \quad [10]$$

Therefore Eq 9 becomes:

$$T_c = -Z\,u + Z\,p = Z\,(p - u) \quad [11]$$

where u is always negative. This means that M decreases with any increase of temperature. On the other hand, p is naturally positive, the distance between the center of gravity and the point of support of the magnet system increasing with rising temperature. Thus, unless compensation is provided for, a symetrically built magnet system will have a positive temperature coefficient.

Temperature Compensation of the New Magnetometers

The modern temperature-compensated magnet system contains two counterweights along the axis of the system (Fig 2). The counterweight near the south pole of the system is on an invar spindle while the one to the north is on an aluminum spindle. In order to keep the system in a horizontal position, the center of gravity of the entire system has to counterbalance the magnetic torque and therefore (in the northern hemisphere) is to the south of the knife edge. With increasing temperature all parts of the system expand symmetrically, with the exception of the two spindles which shift the center of gravity to the north and thus produce a contraction of the value of a and

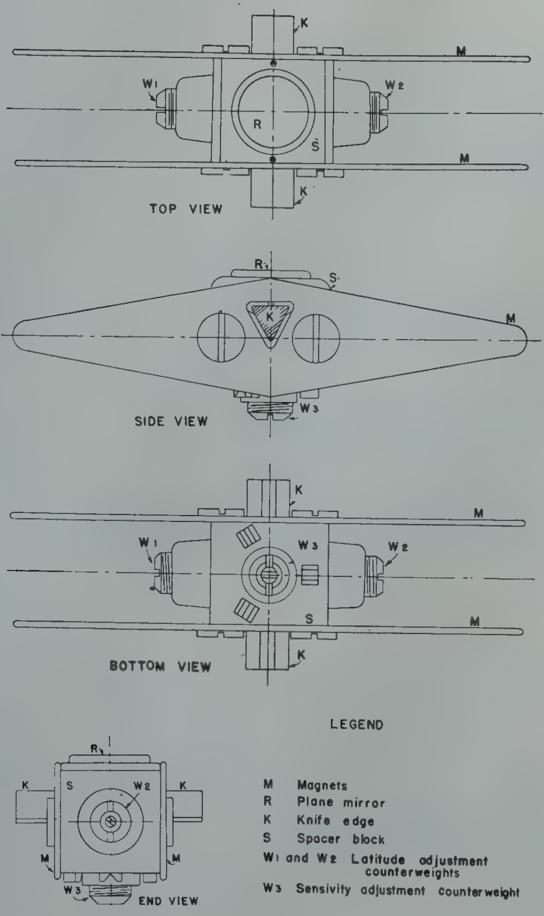

FIG 3—Diagram of old type Askania magnet system which is not temperature compensated.

a negative coefficient p. With appropriate values of the weights *We* and *Wt* and a proper location of *Wt* along the aluminum spindle, a complete temperature compensation can be obtained, the value of p being made negative and equal to the negative value of u:

$$p = u \qquad [12]$$

Temperature Compensation of the Old Magnet System

The theory of this compensation was developed by the author in the spring of 1933, while he was engaged in repairing a magnetometer belonging to the Mines Domaniales de Potasse d'Alsace, in France. It is based on the fact that in the old system the spacer block is made of aluminum which has a coefficient of expansion different from that of the cobalt steel out of which are made the two magnet-blades (see Fig 3).

If an old magnet system is put together with perfect symmetry, it would have in the Ontario-Quebec mining districts of Northern Canada a temperature coefficient of:

$$T_c = Z\,(p - u)$$

where Z would be of 0.58 gauss, p the coefficient of expansion of aluminum 2.4×10^{-5}, u the temperature coefficient of the magnetic moment 14.10^{-5}. Thus, using gammas (1 gamma = 1/100,000 gauss)

$$T_c = 58{,}000\,[2.4 - (-14)]\,10^{-5}$$
$$= 0.58 \times 16.4 = +\,9.5 \text{ gammas}$$

Let us consider now the effect of a nonsymmetrical attachment of a magnet blade to the aluminum spacer block.

The position of the common center of gravity of the whole system depends on the position of each of the centers of gravity of the two blades and the three counterweights. When expansion occurs the relationship may be expressed by the formula:

$$ma_o p = m_1 l_1 v_1 + m_2 l_2 v_2 + m_3 l_3 v_3 + m_4 l_4 v_4 + m_5 l_5 v_5 \qquad [13]$$

where m_1, m_2 . . . m_5 are the respective masses of the various parts of the system, l_1, l_2 . . . l_5 the distances to the centers of gravity or to the points of attachment and v_1 . . . v_5 the coefficients of expansion of the different materials involved. Correspondingly, p will be expressed by a polynomial:

$$p = \frac{m_1 l_1 v_1}{ma_o} + \frac{m_2 l_2 v_2}{ma_o} + \dots + \frac{m_5 l_5 v_5}{ma_o} \qquad [14]$$

If any term of this polynomial becomes sufficiently large and negative so as to match the value of the negative u, the temperature coefficient of the magnetometer will be considerably reduced if not actually annulled.

Let the distance between the axis of the knife edge and the screws holding the magnet blades be $l_b = 0.9$ cm; the coefficient of expansion of aluminum forming the block is:

$$V_a = 2.4 \times 10^{-5}$$

while the coefficient of expansion of the cobalt steel is:

$$V_s = 1.27 \times 10^{-5}$$

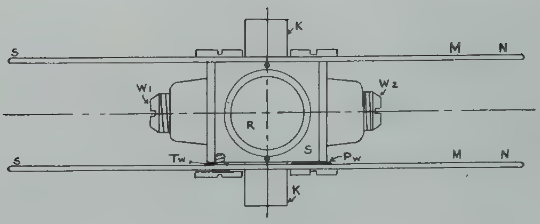

FIG 4—Diagram of old Askania magnetic system partially temperature compensated by attaching one of the magnet blades to aluminum spacer block by means of north screw only.

Legend: *Pw*, paper washer under north screw, which is tightened and holds magnet blade; *Tw*, tubular washer that keeps south screw from pressing magnet blade against block; N-S, showing polarity of magnet blade; for remainder of legend see Fig 3.

finally the mass of each magnet blade is:

$$m_b = 10.8 \text{ g}$$

If one of the blades is to be attached to the aluminum spacer block by the north screw only instead of by both screws which normally press it against the block, any increase of temperature will result in the elongation of the distance between the axis of rotation and the point of attachment, that is, the screw, this according to the coefficient of expansion of aluminum, while the distance between the center of gravity of the blade and the same point of attachment will vary according to the coefficient of expansion of steel.

Thus, an increase of 1°C will shift the center of gravity of the blade towards the north pole of the magnet system for a distance of:

$$(V_a - V_s)\, l_b$$

and the polynomial giving the value of p (Eq 14) will be changed by a term:

$$p' = -\frac{m_1 l_b\,(V_a - V_s)}{m a_o} \qquad [15]$$

The corresponding effect on the value of the temperature coefficient of the magnetometer will be:

$$T_c' = -\frac{Z m_1 l_b\,(V_a - V_s)}{m a_o} \qquad [16]$$

Substituting the value of a_o from Eq 10 we will have the value of the change of the temperature coefficient when a magnet blade is attached by the north screw only instead of by both holding screws:

$$T_c' = -\frac{m_1 g l_b\,(V_a - V_s)}{M_o} \qquad [17]$$

This is the final formula sought by our theory.

It is interesting to note that although the temperature coefficient T_c is function only of Z, u, and p, T_c' is independent of Z but depends on the mass m_1 of the blade and on the magnetic moment M_o of the entire system.

Substituting numerical values of $m_1 = 10.8$ g; $l_b = 0.9$ cm; $V_a - V_s = 1.13 \times 10^{-5}$ we obtain:

for

$M_o = 1500\ \Gamma\ \text{cm}^3 \qquad T_c' = -\ 7.18\Gamma$

$M_o = 1000\ \Gamma\ \text{cm}^3 \qquad T_c' = -10.77\Gamma$

$M_o = 700\ \Gamma\ \text{cm}^3 \qquad T_c' = -15.18\Gamma$

In areas of a high value of Z where T_c is usually around +9.5 gammas it is evident that the loosening of the south screw of one of the blades of the magnet system brings the total value of the temperature coefficient of an old-type magnetometer very close to zero.

Our organization has reduced the temperature coefficients of all its old instruments by this method and, in every case, the experimental measurements of the new temperature coefficient were found to check closely with the predicted values calculated by the formula, Eq 17.

In practice it was found that the best results were obtained if a small paper washer was placed between the blade and the aluminum block under the north screw which was kept tight to hold the magnet blade. The loosened south screw is kept in place and tightened against a small tubular washer pressing against the aluminum block and inserted into the hole of the blade. The tubular washer must be just a shade longer than the thickness of the magnet blade, in order that the blade remain free of the screw (Fig 4).

It is interesting to point out that the fact of leaving only one screw to hold one of the magnet blades decreases the danger of slipping of the blade along the aluminum block which often creates considerable changes of zero point values in old magnetometers. It may be noted that the writer has calculated[4] that temperature changes produced differential stresses between the aluminum block and the magnet blade which were of nearly one kilogram for each degree centigrade.

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Municipal-water Needs vs. Strip Coal Mining

BY GREGORY M. DEXTER

Summary

Recent litigation in Pennsylvania between three coal-mining companies and a private water company resulted in the payment by the coal companies of the equivalent of about $500,000 to buy a new water supply to replace that which had been damaged by acid drainage from strip mining. An investigation of the relevant facts indicates that the water could have been treated by spending about $40,000 on equipment and $15,000 yearly for chemicals, labor, and power until the contamination could be reduced about 80 pct. The water company's supply, furthermore, had deteriorated largely by reason of dry weather before stripping was begun by the coal companies. Payment to the water company should not have exceeded $300,000 and could have been as low as $100,000.

Strip mining of coal is common in Pennsylvania. The most important objection to it in Pennsylvania is generally acid pollution of runoff water. An outline of the methods, therefore, that could be used to reduce such pollution, to compare acid-contaminated waters, and to treat such waters to make them satisfactory for public and industrial use is worth-while.

The watershed of the West Penn Water Co. covers about 4.5 square miles, 20 miles west of Pittsburgh, and has two 90,000,000 gal reservoirs supplying about 1,300,000 gal daily for public and industrial use of which 450,000 gal daily are for two railroads.

The pollution arises from weathered derivatives of typical "Coal Measures" rocks, the principal source being pyrite (and marcasite) in Pittsburgh and Rooster coals.

Springs and seeps on hillsides issue from sandstone and limestone formations intersecting shale or clay land surfaces, or from perched water tables. The two reservoirs create an artificial permanent water table. Soap-hardness tests on three seepage samples taken in a dry summer ranged from 150 to 500 ppm. Water sampled came from aquifers above Pittsburgh and Rooster coals, obviating the possibility of contact with pyrite-containing coal.

Since acid contamination of mine waters is the result of chemical reactions of oxygen and water with pyrite, it will be reduced by the exclusion of either air or water.

The Sunnyhill Coal Co. has applied this principle in its "contouring" method of strip mining by burying contaminated coal-waste material under a compact, smoothed-up surface of topsoil, gravel, clay, and sand at a cost

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of about $250 per acre.

Supplementary methods for controlling localized flow of water include the use of cement grouting, sheet piling, chemical impregnation, tar blankets, and bentonite.

Average rainfall on West Penn watershed is probably 40 in. annually of which about 45 pct is surface and ground runoff. Ground runoff is probably about 25 pct. Surface runoff is increased by "contouring."

Infiltration of rainfall ranging from 0.3 to 2.4 in. per hour depends on porosity and moisture content of the ground. Although oxygen dissolved in the infiltrated water, surface evaporation, diffusion, temperature, wind, and barometric effects are slight in the oxidation of sulphur-bearing material, "contouring" by compaction and increased growth of vegetation is a definite help in its retardation.

Soap hardness of contaminated water is affected by weathering of exposed material, resulting first in increased alumina. Increases of hardness in dry weather (amount is reduced by a swamp on a stream) and decreases with rainfall have been shown repeatedly. No increase in hardness of water in dry weather is an indication of the beneficial effect of "contouring."

Soap-hardness tests on water from the lower reservoir of the West Penn Water Co. showed values as high as 200 ppm in a wet year prior to the beginning of new stripping operations in 1941. In the dry year of 1945, they went as high as 500 ppm, thus showing a deterioration prior to 1941. An approximate relationship of acres stripped to average soap hardness shows that, even if 1½ square miles were stripped on a 4½ square-mile watershed, soap hardness should not exceed 700 ppm.

Neutralization of acids is necessary to eliminate errors in soap-hardness values. Comparison of hardness of various waters is significant only when based on calculation as $CaCO_3$.

Comparisons of contaminated waters using a neutral point of 7.0 are misleading. Electrometric methods should be used to determine pH values. Free acids and mineral acids plus sulphates by "standard methods" are misleading. Total sulphates and total solids may be determined by usual methods. Contamination by sulphates in percentage of total salinity often shows worse conditions in alkaline than in acid waters. Recording gauges on streams and adequate sampling are needed to determine pounds of acid sulphates per day. Coal companies can control contamination by chemical treatment of surface water until "contouring" is completed.

Water softened to 150 ppm with total solids as high as 1000 ppm should be acceptable. Sulphates (SO_4) may be as high as 500 ppm.

Acid-contaminated water can be treated chemically to make it satisfactory for public use by lime and soda-ash softening supplemented by sodium aluminate, sodium silicate, aluminum silicate, and septaphosphate, according to local conditions. Removal of alumina offers no difficulties.

In reservoirs, a variable-level draw-off will reduce contamination.

Treated acid-contaminated water with solids limited to a maximum of 3500 ppm can be used in locomotives if advantage is taken of manual control and continuous blowdown which will range from 10 to 20 pct. Reasonable blowdown values may be had with concentration of solids from 10,000 to 20,000 ppm by using various colloids, organics, and other compounds.

Medical testimony and experience in the West with highly contaminated water have proved that treated water containing 1000 ppm sodium sulphate can be used for drinking.

The conclusion is that "contouring" will reduce acid contamination of water. It permits the economic development of thin beds of coal, where 75 to 95 pct is recovered as compared with 50 to 60 pct in deep coal mining, while protecting water supplies.

Geological data for this paper were furnished by Daniel A. Busch, Geologist, Huntley & Huntley, Pittsburgh, Pa.

Introduction

Strip mining has two principal advantages over underground methods: lower cost and more complete utilization of coal because none need be left for supporting pillars.

Three major objections to it are: (1) the unsightly appearance of the spoil banks, (2) the loss of utility of the land surface, and (3) the acid pollution, in this case of runoff resulting from exposure of coal on the surface. The last objection becomes important where the acid runoff is impounded in a reservoir for use as a water supply.

Attempts have been made to reduce similar pollution of acid mine waters from abandoned mines. Study of a similar attempt applied to strip mining will show that improvement of runoff is also possible.

This paper is an attempt to outline methods that will permit development of some strip mines and yet provide communities involved with acceptable water supplies. Reduction of contamination, errors in determining extent of contamination, and requirements for treatment of contaminated acid raw water for domestic and industrial purposes will be discussed.

The companies involved were the West Penn Water Co., the Sunnyhill Coal Co., and two others.

Location and Drainage of Watershed of the West Penn Water Co.

The watershed of the West Penn Water Co. is an area of approximately 4½ square miles, and is located one mile east of the village of Bavington in Robinson Township, Washington County, Pa. The northeastern portion extends across the county line into Findlay Township, Allegheny County. The region is maturely dissected and has a maximum surface relief of approximately 380 ft.

St. Patrick's Run flows in a southwesterly direction and is the main stream of the area (Fig 1). It has two principal tributaries, both unnamed. One flows parallel to Highway No. 22 and enters St. Patrick's Run, a short distance east of the headwaters of the east branch of the lower of two reservoirs. The second flows south along the west side of the watershed and is impounded where it joins St. Patrick's Run by the lower dam to form the west branch of the V-shaped, lower reservoir. An upper dam also impounds the middle portion of this second stream at a point approximately 2500 ft north of the lower dam.

The West Penn Water Co., therefore, has two reservoirs, each with a capacity of 90,000,000 gal. Acid mine drainage comes mainly from the strip mining of three companies, although some is caused by the abandoned deep mines and old country mines. One of these companies is the Sunnyhill Coal Co. whose operating methods will be discussed. The maximum water consumption is about 1,300,000 gal daily of which about 450,000 gal are used by the locomotives of two railroads. All other water is used principally by the residents of two municipalities.

FIG 1—Map of drainage area of McDonald waterworks.

Geology

TYPES OF ROCK RESPONSIBLE FOR INORGANIC POLLUTION OF STREAMS

The rock strata exposed in this watershed include the upper portion of the Conemaugh and the lower portion of the Monongahela series of the Pennsylvanian system. They are strictly, therefore, a "coal measures" section, consisting of alternating beds of shale, sandy shale, sandstone, limestone, coal, and underclay. The Pittsburgh coal and its Rooster coal are both overlain and underlain by all these types of rock.

Inorganic pollution of the streams comes from the weathered derivatives of these strata. The shales, which make up the thickest portion of the exposed strata, are the principal source of aluminum, manganese, and sodium oxides. The sandstones are probably the main source of silicon dioxide, and the limestones, of the calcium and magnesium salts. Limestone, the most soluble of these several rock types, is the main source of temporary hardness in the raw waters. It is also the source of that portion of the permanent hardness due to calcium and magnesium sulphates. Approximately 10 to 12 ft of limestone occur directly beneath the underclay (1 to 2 ft) beneath the Pittsburgh coal. Limestone occurs, in addition, in all the hills where they extend more than 40 ft above the top of the Rooster coal. This limestone is thin-bedded and locally interbedded with limy shale. Nodular limestone is scattered throughout the shale. This limestone interval has a maximum measured thickness of 25 ft where it caps Bald Knob in the northeast corner of the watershed.

The mineral pyrite (and marcasite) is the principal source of pollution in the reservoir waters. It occurs in the Pittsburgh and Rooster coals and their associated shales.

MODES OF OCCURRENCE OF PYRITE IN COAL

Pyrite (FeS_2) may occur in coal as finely disseminated particles, thin discontinuous sheets, balls, lenses, irregular nodules, and as an extremely thin-vein concentration along inclined cleavage partings. Dark shales and clays associated with coal also contain considerable pyrite. These pyritiferous shales and clays may be the roof-slate material or a part of a "bony" section occurring within the coal itself (compare with Thiessen.)[1]

The pyrite-bearing shales and clays are left in and on the spoil-bank material in the stripping of the coal and are exposed in varying degrees to the atmospheric elements. J. W. Paul[2] says, "The amount of sulphur in bituminous coal ranges from 0.5 to 5 per cent, the average being about 1.50 per cent, approximately half of which is organic and the remainder is in pyrite."

WATER TABLES

Two types of water tables are possible, namely, perched and permanent. In an alternating series of rock formations, such as occur in a "coal measures" section, porous and permeable layers are overlain and underlain by impermeable strata such as shale and clay. Sandstone, because of its porosity, serves in the West Penn watershed as a water-bearing formation (aquifer) where it is overlain and underlain by shale. Likewise, the limestone 40 ft above the top of the Rooster coal frequently contains water, because of its solubility in carbonated waters. Springs and seeps are found frequently where the sandstone and limestone formations crop out on the sides of hills. Such springs coming from a perched water table may flow considerable volumes of water during the winter and rainy seasons and dry up in the summer.

The hardness of such waters varies with the amount of flow and the type of rock serving as an aquifer. One drilled well supplying water from the lime-

[1] References are at the end of the paper.

stone 40 ft above the coal had a hardness of 156 ppm and a pH of 7.1. A spring issuing from sandstone approximately 60 ft below the Pittsburgh coal had a hardness of 280 ppm and a pH of 6.8. Another spring issuing from sandstone approximately 25 ft below the Pittsburgh coal showed a hardness of 500 ppm and a pH of 6.6. In all three instances there is no possibility of the water ever having been in contact with the Pittsburgh or its Rooster coal. The three samples were taken in the late summer of 1945.

Subsurface water will tend to migrate downward where rocks are more or less homogenous in character, until all the pores are filled. The permanent water table is that level below which the rock is completely saturated. It is seldom level, but is a subdued replica of the land surface.

In the absence of a permanent, natural water table, an artificial one has been created by the two reservoirs whose surfaces are at the lowest levels to which the water can flow. Where the permanent table intersects the land surface, seepages occur and there is a flow from the high portions of the water table under the hills into the low areas along the valleys. Since the direction of the flow is never reversed, it is not likely that the permanent water table back under the hills could be contaminated by polluted waters of a reservoir.

Causes of Acid Pollution of Mine Water

Acid contamination varies with numerous factors.[8] Area of coal exhausted is not the sole criterion. Contamination with iron pyrite, for example, may be less at one point than another. Thickness of coal seam, depth of overburden and its character as to permeability, amount of ground water, steepness of surface slopes above the coal seam, dissolved oxygen in water, rate of removal of sulphuric acid and iron sulphate formed, fineness of coal, and exposed surface area of coal and slate with sulphur content are other factors.[4]

The chemical reactions resulting in the pollution of mine runoff waters are complex and only partly understood. The best summary and bibliography is that of Hodge.[4] There are, however, several basic chemical reactions which explain the principal sources of contamination. They involve the oxidation of pyrite (or marcasite). Burke and Downs[5] postulate that pyrite is oxidized as follows:

$$FeS_2 + 3O_2 \rightarrow FeSO_4 + SO_2$$

$$2SO_2 + O_2 + 2H_2O \rightarrow 2H_2SO_4$$

These two reactions, when properly balanced, can be combined as follows:

$$2FeS_2 + 7O_2 + 2H_2O \rightarrow 2FeSO_4 + 2H_2SO_4$$

The following reactions probably occur between ferrous sulphate and sulphuric acid in the presence of additional oxygen[2]

$$4FeSO_4 + 2H_2SO_4 + O_2 \rightarrow 2Fe_2(SO_4)_3 + 2H_2O$$

$$Fe_2(SO_4)_3 + 6H_2O \rightarrow 2Fe(OH)_3 + 3H_2SO_4$$

Morgan[6] suggests that the actual chemical reactions are probably more complicated. These equations show that both oxygen and water are requisites in the production of acid mine waters. In the absence of either, no acid is produced.

Burke and Downs[5] have experimentally demonstrated the extremely slow rate of reaction of the first equation in which pyrite is oxidized to ferrous sulphate and sulphur dioxide. Its rate controls the rate of production of sulphuric acid in mine waters.

Bacteria also may be involved.[6,7,8]

Sayers, Yant, and Leitch[9] show that the passage of moist air over pyrite and marcasite results in a condensate from the moist air which is invariably highly acid. The condensate from moist natural gas which has been in contact with pyrite or marcasite is low in acidity.

Examination of some deep mines that had been sealed showed no sample of water from behind the seals with free sulphuric acid and only one with sulphate of iron in solution.[10] Every sample in open sections of five mines, of long standing, was acid and as a rule highly acid. Three of the mines visited apparently had no acid water either in open or sealed sections. Gas samples from behind seals in those mines generally showed the presence of oxygen to be less than 1 pct.

The practical conclusion, however, is that the exclusion of air or water will reduce acid mine-water pollution.

Reduction of Acid Mine Drainage

BRIEF HISTORY OF MINE-SEALING EFFORTS

The Bureau of Mines, starting in 1925, under the direction of R. D. Leitch, developed the now well-known method of mine sealing. It reduces acid formation by the exclusion of air (oxygen) from the mines in order to prevent the oxidation of pyrite and other sources of sulphur in the presence of water. The sealing of surface cracks and caves reduces the amount of water entering the mines. A complete sealing of abandoned mine entries has not been considered practical, but the draining of mine waters through trapped openings has been shown to be feasible.

A large-scale program of sealing abandoned coal mines was begun in 1933 in West Virginia, Maryland, Ohio, Alabama, Tennessee, and Indiana. This work was carried on jointly by the Ohio River Board of Engineers, the U. S. Public Health Service, several State Health Departments, and Federal relief agencies. It was taken over in 1935 by the Works Progress Administration and the U. S. Public Health Administration and continued for five more years.

A modification of the methods that have been employed in rectifying the pollution of waters draining from deep mines should yield similarly successful results when applied to the waters draining from strip mines. The sealing of coal or shale containing iron sulphide from air (oxygen) will prevent oxidation. Decreasing the amount of water which would normally come into contact with these sources of inorganic pollution should also result in a decrease in the amount of acid formed (Imhoff and Fair,[11] Crichton,[3] LeBosquet and Lyons,[12] Young,[13] Hoak,[14] Meinzer,[15] James,[16] Leitch,[17] Morgan,[6] Babbitt,[18] U. S. Public Health Service,[19,20] and Thiessen[21]).

STRIP-MINING METHODS OF THE SUNNYHILL COAL CO.

The Sunnyhill Coal Co. uses a dragline to remove all the overburden resting on the Rooster coal. The amount removed varies in thickness up to 55 or 60 ft and the width of one cut is approximately 60 ft. The overburden is piled in a long hogback ridge on the downhill side of the cut. The Rooster coal is then loaded out to a depth of about 2½ ft and marketed. The bone and slaty material, which averages 6 ft in thickness and occurs between the bottom of the Rooster coal and the top of the main seam of the Pittsburgh coal, is removed then by a shovel and piled along the base of the spoil bank. The main Pittsburgh vein of coal is then removed and marketed (Fig 2).

The weathered soil, clay, and shale at the top of the undisturbed highwall

FIG 2—Highwall in stripping operations showing Rooster and Pittsburgh coals.

FIG 3—Stripping operation showing spoil material at left and highwall at right.

after the first cut has been completed is bulldozed into the bottom of the first cut in such a manner as to cover completely the loose material from the preceding cut. This blanket of topsoil and clay-shale material is graded with 15 to 16 ton bulldozers, the treads of which work back and forth to build up a compacted cover. This bulldozed fill extends to a foot or so above the top of the Rooster coal in the highwall. The remainder of the overburden removed for the second cut is then piled up as a hogback ridge parallel to the first spoil-bank ridge (Fig 3). The Rooster and Pittsburgh coal seams are removed in the same manner as described for the first cut, and the intervening bone-coal and slaty material is deposited along the base of the second ridge of spoil-bank material (Fig 4).

Successive cuts are made and parallel ridges of spoil-bank material piled up in the manner described until either all the coal of a hill has been removed or the economic limit of overburden is reached. The soil and clay-shale material in the latter case at the top of the highwall are bulldozed into the bottom of the last cut in order to bury the refuse bony coal and slate under a protective blanket of relatively impermeable material. When the last coal of any stripping operation has been removed, bulldozers are used to level off the tops of the spoil banks and fill in undrained valleys and depressions between the ridges (Fig 5). The net result of the bulldozer operation is compacted spoil banks that are contoured for minimum surface exposure and for natural runoff (Fig 6–10). A minimum also of pyrite-bearing coal and shale is on the surface as it has been concentrated in the bottom portions of the spoil banks.

SUPPLEMENTARY PORE-SEALING METHODS

Occasionally stored up water from abandoned mines or springs is released during strip mining. Several methods are available to reduce the amount of such water:

Cement Grouting

Where sand and shale are loose or porous with a small admixture of clay, bore holes may be drilled at intervals and grouted with cement. The seepage from exposed springs or abandoned deep mines need not be stopped entirely; only the level of water should be raised high enough to bury the contaminating material.[22]

FIG 4—Parallel ridges of spoil-bank material in stripping operations.

FIG 5—Top of spoil-bank material before start of contouring.

Sheet Piling

Another method is to drive sheet piling at right angles to the flow of water and grout the joints in the piling.

Chemical Impregnation

A third method is to use the patented Riedel process[23] of chemical impregnation where sodium silicate and calcium chloride in the presence of a catalyst are forced under pressure into the ground. The pores are filled with the resulting calcium silicate and the salt escapes in solution. Although expensive, it might be used to advantage in contoured ground where a clay blanket and other methods outlined above have not been sufficient.

In addition to the preceding, the Shell Oil Co. has developed recently an emulsion of asphalt in water that may be pumped under low pressure into the ground. Chemicals mixed with the emulsion will cause the asphalt to coalesce, producing a mass impermeable to water.

Field Tests

Resort to the methods outlined should not be made until holes have been drilled at right angles to the flow of water and along the center line of maximum flow in order to determine the water table and its slope in various directions. The plane of grouting, sheet piling, or of chemical impregnation can then be determined with some assurance.

Tar

A less expensive method than any of the preceding is to cover the spring or source of seepage with a shallow blanket of sand and gravel whose surface is loose. The surface of this blanket should then be coated with a penetrating application of tar or heavy oil, after which the treated blanket should be buried under additional sand and gravel. This method should be used before contouring is started as otherwise excavation will be necessary to reach the point of application.[24]

Bentonite

The secret of successful contouring is the use of material with a considerable admixture of clay that is bulldozed into place. In the absence of clay, bentonite can be used from the Dakotas at a cost of about $20 per ton, delivered in the Pittsburgh area. The results of permeability tests on such clay-treated soil, shown on Fig 11, suggest that three to four tons per acre of bentonite will produce about the same coefficient of permeability as is obtainable with local clays. The cost of distributing it with fertilizer spreaders and harrowing it in to a depth of 4 to 6 in., using mechanical equipment, should not be great (compare Davis[25] and Turnbull[26]). The U. S. Bureau of Mines has issued a pamphlet[27] that discusses in detail the advantages and limitations of bentonite.

The American Colloid Co. claims that bentonite in the small quantities mentioned above will not retard the growth of vegetation, that the powdered form is preferred, and that the ground should be moistened before applying.

VEGETATIVE COVER

A coal-stripped area contoured in the manner employed by the Sunnyhill Coal Co. can be made to support readily a vegetative cover. By harrowing, fertilizing with lime, seeding with perennial rye grass, and protecting with a temporary mulch of hay, the stripped and restored areas were observed to have a heavy grass cover in less than four weeks.[28]

Peach trees were planted in the spring of 1945 on one of these contoured areas. They survived the following dry summer and in September showed new shoots more than 6 in. long. Toenges[29] reports that poplar, locust, conifers, and walnut needing less care are more commonly planted at a cost of about $20 per acre in 1934 to 1938.

Stripped but uncontoured portions of the watershed of the West Penn Water Co. have remained almost devoid of any vegetative cover since 1932 (16 years). This lack is due to surfaces that consist largely of chunks of sand, shale, coal, coaly shale, and occasionally limestone, all in a more or less unweathered condition. The progressive weathering, in addition, of the pyrite in the coal and shale on the surface results in an acid condition that is not conducive to the growth of most forms of vegetation. The steepness of the slopes of the uncontoured spoil banks also retards the growth of vegetation.

Toenges[30] has noted that in the glaciated regions of the Middle West, where uncontoured spoil banks contain much weathered glacial debris, vegeta-

tion takes root much more quickly. He suggests that for agricultural purposes, exclusive of orchards, the ridges of spoil banks should be leveled off. He places the cost for this leveling at $50 to $60 per acre.

The method of contouring employed by the Sunnyhill Coal Co. not only levels off the ridges and valleys of the spoil-bank area, but smooths the entire land surface into natural runoff slopes with no undrained depressions. The last cut, where a highwall is left intact, is completely filled in so as to leave no visible trace of the position of the highwall. The selective distribution of harrowed sand, shale, and clay, with the addition of humus, results in a surface that will readily support grass and timber. This entire process of restoration was estimated to cost approximately $250 per acre.

Conflict of opinion exists as to the effect of contouring on growth of vegetation. Without contouring there was in this case negligible growth of vegetation; with contouring marked growth of vegetation was obtained.

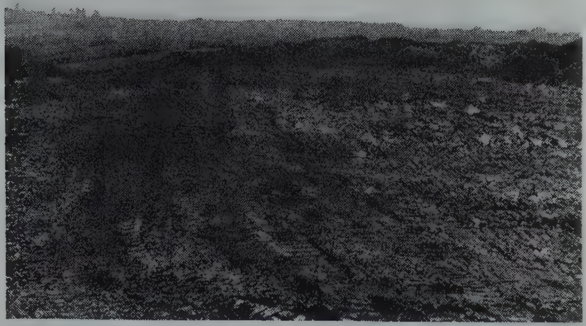

FIG 6—Top of spoil-bank material after some contouring has been done.

FIG 7—Contouring completed showing small peach trees at left.

FIG 8—Contouring in progress showing bottom of last cut covered in.

Size Distribution of Contoured Spoil-bank Material

The surface of a contoured spoil bank consists of a mixture which ranges from very fine clay up to stones a few inches across. The percentage composition of these particles determines the relative density of the mixture. Since almost all the material on the surface of a contoured spoil bank is made up of particles less than 2 in. in diameter, the results of mechanical analyses (distribution of particle sizes) may be readily compared with the equation $p = \frac{(d)^{0.33}}{(D)}$, where p is per cent finer, d is the size of the particle, and D is the size of the largest particle. This equation is the curve representing a fine grading with maximum density of mixture for aggregates having a maximum size from 1 to 2 in., as presented by A. N. Talbot and F. E. Richart.[31]

The mechanical-analysis tests were conducted in accordance with the specifications of the ASTM, Method D 421-39 for the preparation of the sample and Method D 422-39 for the mechanical analyses of the soil samples. The results of these tests are graphically shown in Fig 12 to 15, and are compared with the curve for a fine grading with maximum density of mixture for aggregates having a maximum dimension of 2 in.

Tests discussed above and below were run by Mr. F. Bromilow of the University of Pittsburgh.

Permeability of Contoured Spoil-bank Material

There appears to be a complete dearth of any literature bearing on the permeability of both undisturbed and contoured spoil-pile material. Various investigators have shown conclusively that in soils there is a progressive decrease in permeability, and conversely an increase in moisture content, with increase in depth.

A representative set of surface samples of undisturbed topsoil, undisturbed spoil-pile material, and contoured spoil-pile material were collected at random from various properties of the Sunny-

FIG 9—Completed contouring before planting of grass seed.

FIG 10—Another completed contouring before planting of grass seed.

hill Coal Co. These samples were tested for coefficient of permeability in accord with the methods of Casagrande and Fadum.[32] Subsurface samples, in addition, were taken at various depths and subjected to the same permeability tests. The results of these tests are shown in Table 1.

Surface samples were obtained in steel-pipe cylinders 6 in. in diameter and 6 in. in length, in accord with the recommended method of Hvorslev.[33] The "Falling Head Permeator" method of testing was employed in accordance with the recommendation of Casagrande and Fadum.[32] The results of these tests superimposed on the soil classification of Casagrande and Fadum are shown in Table 1.

The permeabilities of gravel, sand, silt, and clay are given in terms of a coefficient of permeability, which is expressed by Darcy's empirical law:

$$Q = k \cdot i \cdot A \cdot t$$

Q is the total quantity of water flowing through a material of cross sectional area, A, under a hydraulic gradient, i, in time, t. The coefficient, k, called the coefficient of permeability, is dependent not only on the size, shape, and structure of the particles, but also on the void ratio and temperature.

Nos. 1, 2, 3, and 4, of Table 1, represent samples of undisturbed topsoil, all of which fall well within the range of "impervious soils." This is the type of material that the Sunnyhill Coal Co. bulldozed from the highwall, spread, and compacted over the bone and waste-coal material placed in the bottoms of successive cuts. Nos. 5, 6, and 7 represent undisturbed spoil-pile material. No. 7 is the only one of the three falling within the range of "impervious soils." The samples obtained from contoured spoil-pile material are Nos. 8, 9, 10, 11, 12, 13, 14, and 15. All these samples, except No. 8, fall within the range of "impervious soils." Sample No. 8 was taken at a point directly above the only bad-surface seep of water coming from any of the area contoured. Further work by way of grouting, chemical impregnation, or other treatment will be necessary to remedy this situation.

In an effort to determine whether the impervious character of the topmost 6 in. of a contoured spoil pile is merely a surface phenomenon, samples of undisturbed spoil-pile material were then taken at various depths. This sampling was by drilling several auger holes, 4 in.

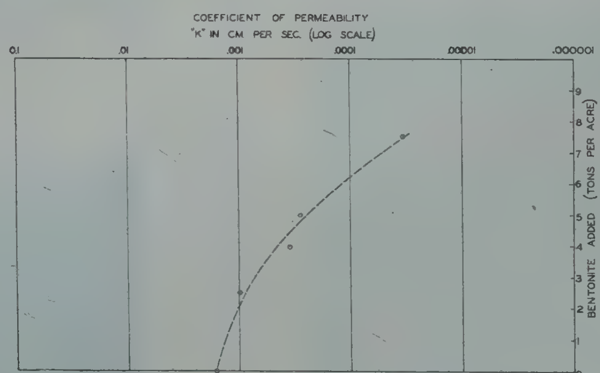

FIG 11—Decrease in *permeability* of *spoil-pile material* to which *bentonite* has been added.

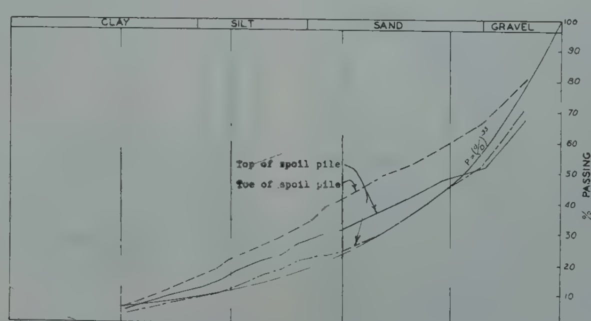

FIG 12—Percentage composition of *contoured spoil-pile material* compared with curve for fine grading with maximum density of mixture for aggregate having maximum dimension of 2 in., location *A*.

Table 1 . . . Results of Permeability Tests Made on Samples of Topsoil and Spoil-pile Material Superimposed on the Soil Classification of Casagrande and Fadum

Coefficient of Permeability "K" in cm per sec (log scale): 100 — 10 — 1.0 — 0.1 — 0.01 — 0.001 — 0.0001 — 0.00001 — 0.000001 — 0.0000001 — 0.00000001 — 0.000000001

Application in Earth Dams and Dikes	Pervious Sections of Dams and Dikes	Impervious Sections of EarthDams and Dikes
Undisturbed topsoil, surface	1	2 3 4
Undisturbed spoil-piles, surface	5 6 7	
Contoured spoil-piles, surface	8 9 10 11 12	13 14 15
Bulldozed backfill, subsurface	B	G J C D E H K F I

Types of soil			
Clean gravel	Clean sands Clean sand and gravel mixtures	Very fine sands, organic and inorganic silts; mixtures of sand, silt, and clay; glacial till; stratified clay deposits; and so on.	"Impervious" soils, eg. homogeneous clays below zone of weathering
		Impervious soils which are modified by the effects of vegetation and weathering	

← (K = 0.0014) = 1 in. per hour through 6 in. thickness water head of 3 in.

Hole 1		Hole 2		Hole 3	
Sample	Depth	Sample	Depth	Sample	Depth
A	2 ft 0 in. lost	*E*	2 ft 10 in.	*G*	2 ft 10 in.
B	6 ft 3 in.	*F*	9 ft 7 in.	*H*	9 ft 7 in.
C	13 ft 0 in.			*I*	19 ft 8 in.
D	19 ft 9 in.			*J*	22 ft 0 in.
				K	29 ft 0 in.

in diameter. Drilling was interrupted at various depths and samples of undisturbed material obtained by driving steel pipe into the undisturbed material. The pipe was assembled by screwing a 6-in. nipple, 2½ in. in diameter, to the lower end of a 5-ft length of 2-in. pipe. Additional 5-ft joints of pipe were added as the depths of the holes increased. The individual 6-in. nipples in which the samples were taken, thus, served as the containers for the laboratory tests for permeability. The open ends of the containers were tightly wrapped in canvas and sealed with paraffin immediately after the samples were lifted out of the hole.

The results of permeability tests run on samples taken at variable depths show that, with the exception of sample *B*, Table 1, all samples fall well within the coefficient of permeability range of "impervious sections of earth dams and dikes."

These results suggest that there is a rapid increase in impermeability of contoured spoil-pile material within the topmost 2 ft. There is, likewise, a progressive impermeability gradient with depth. The abrupt increase in impermeability within a few feet of the surface is explained partly by the filtering out of the fine suspended particles carried through with the first water to fall on the contoured spoil pile. Bulldozing, weight of the overburden, and filtering of fine suspended particles combine to make a contoured spoil pile an effective seal over the pyrite-containing rock underneath.

Meteorological Factors

The role of meteorological factors in the oxidation of sulphur-bearing minerals, occurring in both uncontoured and contoured spoil banks, is frequently overemphasized. The climatic factors that must be considered in the matter of air (oxygen) exchange and resultant oxidation of pyrite are rainfall, temperature, barometric pressure, and wind. Some of these factors are continuously operative while others are only intermittent in their effects.

RAINFALL

Rainfall may run off, infiltrate, or evaporate, and frequently does all three. Romell[34] estimates, however, that rainfall accounts for only one-twelfth to one-sixteenth of the normal aeration of soils.

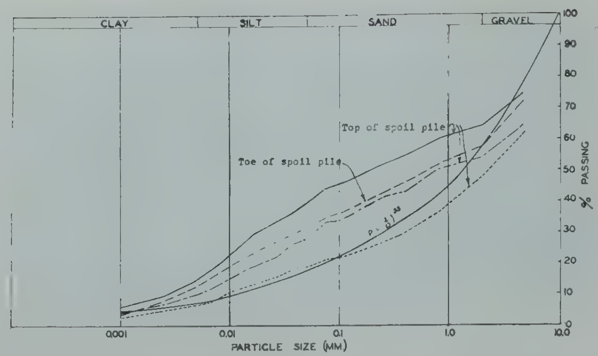

FIG 13—Percentage composition of *contoured spoil-pile material* **compared with curve for fine grading with maximum density of mixture for aggregate having maximum dimension of 2 in., location** *B*.

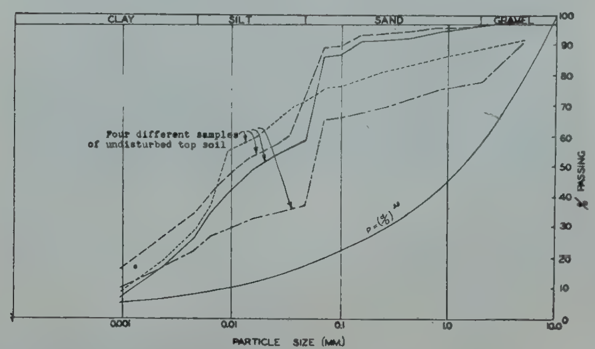

FIG 14—Percentage composition of *undisturbed topsoil material* **compared with curve for fine grading with maximum density of mixture for aggregate having maximum dimension of 2 in., various locations.**

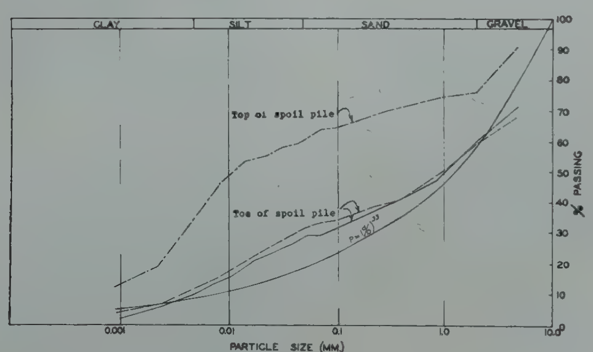

FIG 15—Percentage composition of *contoured spoil-pile material* **compared with curve for fine grading with maximum density of mixture for aggregate having maximum dimension of 2 in., location** *C*.

Rain Gauges

Accurate rainfall data should be obtained by observing well-known rules in the location of gauges. More than one gauge is desirable for slopes as short as 150 to 600 ft. A gauge to record intensity of rainfall is also desirable. Gauges only 1500 ft apart have shown differences of 5 pct in monthly totals with greater discrepancies on single storms or thunder storms.

A gauge generally should not be located at the crest of a ravine or in the lee of a ridge at right angles to the direction of rain-bearing winds which are east or southeast to southwest for northeastern United States. Errors will be caused by air currents in ravines running north and south, eddies with vertical or horizontal axes, adjacent trees, fences, or slopes. Gauges should be placed at least 100 ft away from steep slopes as otherwise the error introduced could be as much as 15 pct. Gauges should be vertical since an inclination of 15° means an error of 3.4 pct. Errors of course can be cumulative (see Flinn, Weston, and Bogert,[35] Horton,[36] and Mead.[37])

Runoff Measurements

A portable weir is unsatisfactory for runoff measurements because the small drainage area involved cannot be measured with sufficient exactness, and the water backing up behind it will seep underneath. Depending on the slope of the ground, the minimum drainage area tested should be from one to several acres. The weir should be substantially built and the upstream side should have a blanket of clay to prevent seepage loss.

Factors in Runoff

Runoff is reduced by surface irregularities, dense vegetation, surface mulches, and inversely, by degree of slope. Some army engineers use the comparative factors for runoff given in Table 2.

Table 2 . . . Estimated Runoff from Various Surfaces

	Per Cent
Asphalt pavement (good)	85 to 90
Concrete pavements with filled joints	75 to 85
Compacted bare ground	*60 to 65
Cultivated ground	*40 to 50
Thin sod cover	*30 to 40
Anchored mulch (2 tons per acre)	*10 to 20
Dense sod cover	*5 to 10

* Depends on degree of slope, character of surface, and sub-soil (see Morrish,[38] Babbitt,[39] Metcalf and Eddy,[40] Meinzer,[41] Mead,[42] and von Schon.[43]

Mead[44] says an area with clay soil will discharge much more than a district with sandy soil. Discharge from more rolling and less sandy land is estimated at 50 pct more than from flat, alluvial land, but for clay soils the quantity is doubled, and for slope tripled as compared with flat land in determining flood runoff. Meyer[45] says also that clay soils facilitate runoff by capillary action bringing moisture to the surface for evaporation, that the steeper the slope of a watershed, the greater the runoff, and that swamps and marshes, while tending to retard

and equalize ordinary runoff, greatly reduce the quantity of water reaching streams (compare Hardenbergh[46]).

Contouring in strip mining puts clay on the surface, increases the general slope by eliminating depressions, and thus increases surface runoff.

Surface and topographic conditions determine the time available for infiltration. The longer the infiltration, the less the runoff. One test showed that the runoff percentage increased up to 10 min on bare land and up to 50 min on forested land. Increase of porosity vertically and increase in organic matter mean greater infiltration and less runoff (Meinzer,[47] Neal,[48] and Musgrave[49]). Contouring in strip mining decreases porosity and thus increases surface runoff.

During the period when tests were being made to determine the relative merits of contouring as carried out by the Sunnyhill Coal Co., there was little rain; hence significant data in this area are lacking.

Amount of Ground Water

A continuous series of runoff tests were made at Chartiers Creek from 1921 to 1933.[50] It has an approximate drainage area of 260 square miles and is located in Washington County, Pa., approximately 15 miles southeast of the West Penn watershed. During this period of 13 years, the average annual precipitation was 39.5 in. and the average runoff was 17.7 in. Thus, an average of 44.8 pct of the precipitation falling on this drainage area became runoff (surface and ground). The rock strata in this area comprise a typical "coal measures" section and the soil is mantled with a vegetative cover.

The preceding surface and ground runoff of about 45 pct is proof that the ground runoff for such an area as the West Penn watershed is much less. Data on this point are confirmed by Hoyt,[51] Meinzer,[52] and Carpenter.[7]

Crichton[3] concludes that probably not over 25 pct of the rainfall appears as mine drainage or ground runoff in deep mining. Contouring, with its striving for steep surface and highly impermeable slopes, is an attempt to decrease the amount of ground runoff by increasing the surface runoff.

This approach of the Sunnyhill Coal Co. is supported by the following comment:

Reduction in acid mine drainage from deep mines by sealing to reduce ground runoff has been studied by the U. S. Public Health Service.[20] It found for 100 sealed mines in West Virginia that the total acid discharged was reduced about 50 pct in a few months after sealing and about 80 pct in 4 years. Such reduction, significantly, was obtained mainly by reducing the flow of drainage water or ground runoff from the mines by about 70 pct at the end of 4 years.

Infiltration

The infiltration factor is the amount of water the ground absorbs in its pores. This amount can be so great after a long, dry spell and with a light rain or a short, intense thunderstorm as to reduce the runoff to negligible proportions (compare Neal[48]). That portion of the rainfall which percolates down through the voids of a contoured spoil bank is dependent upon the number and size of the pores, the amount of rainfall, the nature of the vegetative cover, and the angularity of the surface slope. Some of this moisture returns to the surface as seeps along the lateral margin of the spoil bank, but as time progresses, the volume of seepage should decrease. Some of this soaked-in water adheres hygroscopically to the surface of soil particles. Small quantities undoubtedly become water of crystallization on various minerals. A part returns to the surface through capillary action and escapes to the atmosphere by evaporation. The surface tension of water for the enclosing surfaces of minute voids is such that this water never escapes. In fine textured soils under conditions of compaction, such as characterizes much of the material of contoured spoil banks, a marked decrease in the number and size of such pores due to water content is an extremely effective factor in retarding the rate of air exchange; oxidation is thereby reduced.

Baver[53] says, "Infiltration of rainfall into the soil may cause a renewal of soil air in two ways, namely, the displacement of air in the pores by the water, which is subsequently displaced again with air, and the carrying in of dissolved oxygen in the water."

A contoured spoil bank is composed of compacted material containing a sufficiently high percentage of fine materials (discussed under "Effects of Diffusion") to prevent any such great exchange of soil air. A considerable amount of entrapped air remains which is not forced out by infiltered water. What little renewal of air does occur as a result of infiltration is governed by the periodicity of the rainfall and the permeability of the soil material.

The amount of dissolved oxygen in infiltered water is relatively insignificant from the standpoint of oxidation of pyrite in a spoil bank. According to Steel[54] the solubility of oxygen in fresh water at 51.8°F is 11.08 ppm and at 60.8°F is 9.95 ppm. These solubilities are maximum values for the temperatures given. Temperature measurements of the ground water in one of the contoured areas of the Sunnyhill Coal Co. range from 52° to 60°F.

Porosity as Affecting Infiltration

Some approximate percentage of porosity variation with different soils will show how the infiltration factor is affected. Dixey[55] furnishes the following data: A collection of perfect spheres of solid matter will have porosity percentages running from about 26 to 48 depending on the size of the spheres. The percentage for clean gravel will be from 32 to 36, for sandy soils from 30 to 35, and for clayey loams about 45. The addition of 3 pct of loam to sand will reduce the porosity of the sand to one-tenth of its former value. A clay will absorb up to 52 pct of its weight in water, but such a mass does not yield water as it is so tightly held by capillarity that little or none of it can escape. Contouring is an attempt to reduce porosity.

Evaporation

A small proportion of rainfall striking the surface of a spoil bank evaporates immediately; the rate is governed by the degree of humidity and the temperature of the atmosphere. During periods of prolonged drought, the surface portions of a contoured spoil bank will gradually give up moisture content. In a humid region, however, this evaporation is merely a shallow surface phenomenon. Contouring will have little effect on evaporation except through the action of vegetation.

TEMPERATURE EFFECTS

Expansion and contraction of the air within the pore spaces of a soil, together with the tendency for warmer air to rise, may cause some exchange of atmosphere. Romell[34] considers temperature differences between the soil and the atmosphere to be responsible for not more than $\frac{1}{240}$ to $\frac{1}{480}$ of normal aeration. Contouring by reducing

porosity will reduce temperature effect.

BAROMETRIC-PRESSURE EFFECTS

An increase or decrease in barometric pressure should result in a small amount of compaction or expansion of the air within the soil, causing a rinsing action, but only if the atmospheric air had penetrated the pores in the upper portion. Barometric pressure changes, however, are so slight over any region as to be of little significance in affecting soil air.[53]

Contouring by compaction reduces the access of air.

WIND EFFECTS

Air in motion and in contact with a land surface will exert a slight suction upon the soil air. The depth effect of this pressure differential would be a small fraction of an inch. Winds of high velocity might exert a slight pressure on soil air. Romell says wind action could not be responsible for more than about 1/1000 of the normal aeration on vegetated soils (compare Baver[53]). Although this value may be higher for bare, porous, unprotected soils, it is a relatively small part of total aeration.

Contouring by permitting vegetation reduces wind action on the soil.

SUMMARY OF METEOROLOGICAL EFFECTS

Most of the data available indicate that meteorological factors of rainfall, temperature, barometric pressure, and wind are of little consequence in effecting an exchange of soil air. Perhaps one-tenth of normal aeration of soils may be assigned to this combined effect. Contouring in strip mining keeps this effect at a minimum.

Effects of Diffusion

Diffusion as applied to gaseous exchange in soils is by far the most effective means by which the atmospheric gases can aerate soils under normal conditions. It is a continuous process as long as there is any free pore space in the soil. Both compaction and water saturation are operative in contoured spoil banks to decrease the amount of free pore space.

Diffusion can be reduced to almost nothing by 80 pct water saturation. High water-holding capacity of fine textured soils will lower the diffusion coefficient to about one half that of free diffusion. A compacted area at the immediate surface materially restricts diffusion (Baver,[56] Buckingham,[57] and Hannen[58]).

The Ohio River Pollution Survey Report[20] on "Acid Mine Drainage Studies" includes a graphic diagram of the theoretical relationship between oxygen concentration in mine air and the rate of acid formation. The deduction may be drawn from this diagram that a 25 pct reduction in oxygen reduces the relative rate of acid-production reaction 57 pct; a 50 pct reduction in oxygen reduces the reaction rate 87 pct; and a 75 pct reduction in oxygen reduces the reaction rate 98 pct.

These theoretical considerations, when applied to the various factors in the reduction of pore space in contoured spoil banks, lead to the conclusion that diffusion is of little consequence.

Factors Affecting Soap Hardness

WEATHERING

The first effect of contouring is to expose considerable unweathered material so that heavy rains can be expected to wash down considerable mineral matter. This result is shown on one stream where the soap hardness had been as high as 5000 ppm in the summer of 1944 as the result of stripping operations. This value dropped to 700 to 2500 ppm in 1945. The first effect of contouring started in September 1945 was to drive the soap hardness down to about 500 ppm, but a heavy rain drove it up to about 1500 by washing down exposed, unweathered mineral matter.

The great source of aluminum compounds is the feldspars, mica, clays, associated minerals, and numerous phosphates and sulphates.[59] The shales contain aluminum silicates as their most important constituent. Fitch[59] says that the decomposition of these shales, in the presence of pyrite, results in the formation of soluble sulphates that find their way into streams. Another source of alumina in water is the alums. When sulphides of metals oxidize, sulphuric acid is formed and this acts on the aluminous rocks to form aluminum sulphate. The preceding, of course, explains the claim of the water company that the alumina content of its water supply had increased greatly. Such increase was probably due more to disturbance of the surface clay and shales by stripping than to acid mine drainage.

SWAMPS

The record of soap hardness on one stream, where stripping operations without contouring by another coal company had been extensive, showed no increase during June, July, and August 1945, but a heavy rainfall in September caused a marked increase from about 400 to 900 ppm. The flow of water during the summer months had been essentially uniform and substantial. These results follow logically from the large swamp area on this stream above the sampling point. Von Schon[43] has shown the effect of the character of drainage area upon runoff volume. A typical comparison for one condition of ground-storage depletion of water is given in Table 3.

Table 3 . . . Typical Comparison for One Condition of Ground-storage Depletion of Water

Type of Drainage Area	Total Depletion in Ground Storage in Inches of Runoff	Corresponding Monthly Flow in Inches Runoff
Bold relief and in highlands	3.0	0.3
Drift-covered rock and no swamp storage	3.0	0.75
Sandy watershed with large swamp storage	3.0	1.35

We may conclude from the preceding that on this stream the swamp storage acted to assure uniform flow and dilute acid contamination until a heavy rainfall washed in contamination in excess of dilution available (see also Mead[60]).

HIGHLANDS

The record of soap hardness on another stream showed an increase from a March-and-April value of about 700 to about 1500 in June, with no increase in July and August 1945. Rains in September drove it down until a value as low as 250 ppm was reached. This stream had negligible swamp storage and its flow dropped to as little as 1 gpm in August and September. The Sunnyhill Coal Co. was carrying on extensive stripping and contouring operations during 1945 on tributaries in the headwaters of this stream. The lack of increase in the soap hardness during the dry summer months and the drop in amount of soap hardness with increase in rainfall are evidence of the

effectiveness of the contouring. The contrast in results as to soap hardness between this stream and the preceding where there was considerable swamp drainage is significant as to the benefits from contouring.

Dry Weather

Soap-hardness records were kept by the water company on various streams for periods ranging from 4 to 12 months. Samples were taken at intervals ranging from one week to one month. Charts on five of these streams in which the Sunnyhill Coal Co. was interested showed marked increase in soap hardness during the months of June, July, and August 1945. Typical increases in ppm were as follows: 300 to 800, 300 to 700, 200 to 1100, 700 to 1000, and 1200 to 2000. Such increases should have been expected as study of the U. S. Weather Bureau records at four stations east, south, west, and north of the watershed showed the rainfall to be from ½ to 1 in. less per month than the long-term average for each of those three months.

This conclusion has been supported by studies of acidity and dissolved solids in the Allegheny, Monongahela, Youghiogheny, and Delaware Rivers (compare Clarke,[61] Powell,[62] and Dixey.[63]

SPOT SAMPLING

Spot samples are of limited value in determining the mineral content of a stream. After a drought of two or three weeks, soap hardness values will be considerably higher than after a moderate rainfall. A heavy rainfall or a thaw in the spring will wash down mineral matter that will cause an increase for a day or so in the soap hardness, followed by a sharp drop. A slight rainfall of less than a quarter of an inch will usually have little effect unless the ground is frozen. A moderate rainfall of half an inch or so will cause reduction in soap hardness within 24 hr.

The preceding statements are supported by a study of the record of soap hardness for the raw water from the lower reservoir of the water company. In general, the soap hardness dropped when the rainfall exceeded 1 in. in one day, but a fall of ½ in. or less appeared to have little effect except in winter months when the ground was probably frozen. In some cases, between 1937 and when new stripping started in 1941, the soap hardness increased in the spring in spite of considerable rain. This increase was probably due to stratification of water in the reservoir, with the turnover due to warmer weather, and the effect of stripping operations. The relationship between runoff and rainfall at best is complicated as may be shown by reference to Baver[53] and Neal.[48]

Table 4 . . . Effect of Stripping Operations on Contamination of Reservoirs

Values of Soap Hardness, PPM					Range in Year, PPM	Yearly Average, PPM	Weather Comments
Year	Low and Month		High and Month				
No stripping since 1932							
1937	74	Jan. May	128	Dec.	54	101	Wet year, about 5 in. above average.
1938	78	Mar. Apr.	204	Dec.	126	135	About average rainfall for year.
1939	120	Feb.	172	Sept.	52	151	Dry year, 4 in. below average.
1940	92	Apr.	188	Nov.	96	135	Wet year, 4 in. above average. For West Va., wettest year since 1937. Note, however, that for 6 mo. of min. rainfall, the total was 2 in. less than average.
New stripping began 1941							
1941	112	Jan.	180	May Nov.	68	151	For Pa., only twice before have lower amounts of precipitation been recorded. For West Va., fifth driest year of record.
1942	92	Mar.	184	Nov.	92	152	For Pa., the outstanding feature was the frequency and extent of flood-producing storms. For West Va., the mean annual precipitation seventh greatest of record.
1943	124	Jan.	270	Dec.	146	192	For Pa., from an unusually wet May and Oct. to the driest Sept. of record. The total precipitation was about 4 in. below normal. For West Va., drought conditions began in the northeastern panhandle division and continued until end of Sept.
1944	156	Mar.	432	Nov.	276	257	For Pa., driest summer since 1932. For West Va., from June through August, drought of unusual severity prevailed.
1945 to end August.	252	Mar.	488	Aug.	236	394	Precipitation in June, July, and August from 0.5 to 1 in. less than average for each month.
Increases	252−120 = 132		488−204 = 284		276−126 = 150	394−151 = 243	

Note: Comments on weather conditions taken from Annual Climatological Summary for Pennsylvania and West Virginia of the U. S. Weather Bureau for the various years. Rainfall figures are based on records of four weather-bureau stations that are east, south, west, and north of the watershed in question.

CONTAMINATION IN SHEETWASH

The preceding explanations of weathering and effect of dry weather are supported by soap-hardness tests on samples of sheetwash. Samples taken the morning following a severe thunderstorm after a long dry spell showed soap-hardness values of 1000 to 1500 ppm. These values dropped after a storm a few days later to 100 to 300 ppm. Mason and Buswell[64] report in contrast that one test showed the total solids in rain water to be about 30 ppm and the average hardness 3 ppm. The comparative figures for upland surface water were about 100 and 54 ppm, respectively. These values, of course, would vary with the locality where the rain-water and surface-water samples were taken.

The data above suggest that a fairer test of the effectiveness of contouring is the relative change in the amount of alumina and iron oxide in the raw water supply. The immediate result of contouring is to increase the alumina, but as weathering of the final surface slows down the alumina should drop. Probably three years would be necessary to get the maximum drop in alumina. At the same time, the iron oxide in the water should decrease as the contouring becomes effective in reducing the amount of water in contact with the pyrite, marcasite, and similar contaminants in the carbonaceous shale.

EVIDENCE ON EFFECT OF WEATHER

Increased contamination of reservoirs from acid mine drainage was not entirely due to the new stripping operations that began in 1941 as Table 4 from water-company records shows.

An almost straight-line relationship between decreasing yearly rainfall and increasing average soap hardness of the reservoir water was found for the years 1937 to 1939. The yearly rainfall used was the average of the four

weather-bureau stations mentioned above. Increased rainfall in 1940 reduced the average soap hardness. The conclusion is reasonable, therefore, that dry weather caused the increase of average soap hardness from about 100 to about 150 ppm. This conclusion is in harmony with other studies of the effect of dry weather on hardness of water.

Rainfall in 1941, when the new stripping started, was about the same as in 1939 and the soap hardness was also about the same. Increased rainfall in 1942, about the same as in 1940, kept the soap hardness about as low as in 1939 and 1940. This result suggests that increased contamination was first felt in 1942. But in 1944 with about the same rainfall as in 1940 and 1942, the average soap hardness had doubled.

The preceding indicates that the dry summer of 1945 undoubtedly had marked effect in increasing the average soap hardness for the first eight months of the year to 394 ppm.

ACRES STRIPPED

The water-company's records for soap hardness of its raw water covered for the period 1937 to 1945 inclusive, were based on samples taken from the raw-water pipe in the filter plant and, as indicated later, were probably higher than reasonable due to probable stratification in the lower reservoir during summer and fall months. The maximum soap hardness had increased from 100 to 200 ppm for 1937 to 1941 inclusive to about 500 ppm for the summer of 1945.

The water company claimed that additional strip mining would increase the hardness still further. A chart by years on rectangular coordinates showed the growth of acres stripped and of soap hardness.

The information furnished was plotted on semilog paper to get trends. Then using average, yearly soap-hardness figures in conjunction with acres stripped on log-log paper, it is possible to show that 1000 acres stripped would only develop about 700 ppm of soap hardness. But 1000 acres stripped or 1½ square miles is probably the maximum for the 4½ square miles of watershed involved. Any relationship that exists is at best, of course, very rough as the estimates of acres stripped are not based on detailed surveys. It also erroneously assumes that nothing can be done to reduce contamination.

An even greater error was introduced in neglecting the effect of rainfall on the increasing hardness of the raw water. The months of June, July, and August 1945, for example, showed about 0.5, 1.0, and 0.9 in. respectively, less than the long-term average of four Weather Bureau stations located north, east, south, and west of the watershed.

The preceding suggests that the completion of strip mining on the watershed would not have greatly increased the contamination of the water company's reservoirs. Any relationship existing between acres stripped and soap hardness is obviously so complicated that study of it is only possible on logarithmic paper.

CONTOURING

An example of the effectiveness of contouring is shown by a stream where soap-hardness values in the winter of 1944 to 1945 ranged from 1000 to 5000 ppm. Soap hardness during the summer of 1945, in spite of the dry weather, decreased from 1500 to 1000 ppm and in September was as low as 700 ppm.

An even more impressive example of the improvement from contouring is found on a stream where the soap hardness in the winter of 1944 to 1945 ranged from 1000 to 9000. During the summer months of 1945 it dropped from 7000 to 4000 and by September was down to 1000 to 2000 ppm. The flow on this stream, however, was contaminated by the discharge from an old country mine and by a large spring that had been exposed in a stripping operation. Both flows can be controlled as outlined previously under supplementary pore-sealing methods. Contamination on this stream, therefore, can also be reduced to negligible proportions.

The best example of the benefits of contouring is found on one farm where all stripping operations have been completed and contouring has been effective since the spring of 1945. Soap hardness dropped on one stream from this farm from about 1000 ppm in March and April to 500 in the dry summer months and to about 200 in September. Spoil-bank samples showed soap hardness as high as 5500 ppm in April. Soap hardness did not increase during the dry summer months.

Still another stream from this same farm showed soap hardness ranging between 500 and 800 ppm during the summer months, with no sign of any increase. In September the soap hardness was about 500 ppm and showed little variation with rainfall. The conclusion again is reasonable that contouring reduces the soap hardness values in acid-contaminated streams from several thousand to less than 1000 ppm.

Additional proof of the effectiveness of contouring will be found in the disposal of refuse or garbage in sanitary fills at San Francisco.[65] Refuse fills of 4 ft are covered with 2 ft fills of earth and rock. Test pits dug after ten years showed garbage with very little sign of the decomposition that is possible with air and water. Fruit and vegetables were so fresh that there was little loss in color. Newspapers were readable. Tin cans were bright and without rust. Metal had deteriorated from electrolytic or galvanic action but had no rust.

ERRORS IN TESTS FOR SOAP HARDNESS

Soap hardness is the test that water chemists are likely to use in comparing acid-contaminated waters. It leads, however, to misleading results, particularly where the concentration of contamination is high.

Kean and Gustafson[66] show that the soap hardness is directly proportional to the quantity of acid added to distilled water. The error in the test even for normal waters, according to Mason and Buswell,[67] is 10 pct. Dr. Hale[68] limits the accuracy to 10 to 15 ppm on water below 300 ppm of hardness, but acid-contaminated water can easily run as high as 10,000 ppm. Scott[68] says the American Public Health Association has recommended against the soap-hardness test because of unsatisfactory results. Variation in the amount of calcium and magnesium in different waters and the presence of sodium salts such as sodium sulphate, iron and aluminum salts, free mineral acid, and some forms of silica prevent precision in the soap-hardness test (Solvay Process Co.,[69] Betz and Betz,[70] Suckling,[71] Hoover,[72] and Betz and Betz.[73]

An attempt to compare waters, varying greatly in their contamination, by using the soap-hardness test is certain to be misleading.

METHOD OF NEUTRALIZING ACID-CONTAMINATED WATER

"Standard Methods"[74] requires that any water showing acidity below pH 4.4 shall be neutralized with alkali to methyl orange. Failure to follow this procedure may cause an

error of 10 to 15 pct. For example, a soap-hardness value on unneutralized acid water of 2700 ppm might become 2300 ppm when neutralized to methyl orange. Betz and Betz[70] agree that free mineral acid must be neutralized previous to the soap-hardness test. Acid water, according to Solvay,[69] should be neutral to methyl orange indicator by titration with sodium hydroxide before the soap test.

For acid waters, neutralization with sodium carbonate until phenolphthalein indicator shows pink is not safe; it should be used only on neutral waters.[75] The results of determining total acidity of water with phenolphthalein indicator in the presence of aluminum sulphate or any other salt of a similar type are valueless as the pink color does not appear until an excessive amount of sodium hydroxide has been added; reactions take place slowly.[76]

Acid-contaminated waters contain large amounts of dissolved salts that act as buffers. Hopkins[77] has explained that adding 0.01 N hydrochloric acid to a liter of pure water at pH 7.0 will change the pH to about 5.0. This same amount of acid added to water containing dissolved salts or buffers will cause a pH higher than 5.0. Every substance in the solution will combine with the acid and thus decrease the number of hydrogen ions set free. This buffer action permits the addition of considerable amounts of alkali to acid-contaminated waters without changing the pH value. Such addition of alkali is required in the soap-hardness test. The greater the amount of dissolved material the greater the error in the determination of pH at the end point for neutralization. An attempt, therefore, to compare by soap-hardness values different acid-contaminated waters where free acid has to be neutralized can end only in misleading conclusions.

In spite of the preceding, some chemists have argued for neutralization of acid water to pH of 7. Even greater errors are introduced by this procedure. A study of about 100 comparisons showed a variation in soap hardness from a few per cent to as much as 50 pct. For example, a soap hardness of 7000 ppm might be as low as 4500 ppm if the free acid is neutralized to pH value of 7.

Total hardness should be determined from amounts present of silica, iron, aluminum, calcium, magnesium, and manganese by calculation as $CaCO_3$ instead of by using soap-hardness values.

As will be obvious from the preceding, soap hardness is not a satisfactory method of determining the permanent or sulphate hardness. A comparison of soap hardness and total hardness as $CaCO_3$ shows for several samples reasonable agreement up to about 250 ppm and increasing divergence as the soap hardness increased to 1000 ppm or over. This divergence is increased by free sulphuric acid and other contaminants, the amounts of which will vary greatly in acid mine waters. It makes impossible the fair comparison of acid mine drainage by soap-hardness tests.

Determination of the Amount of Acid Contamination of Raw Water

LOCATION OF NEUTRAL POINT IN WATER

Too much insistence is placed by water chemists on the requirement that satisfactory water for municipal purposes must have a pH of 7 or higher. Morgan[6] has pointed out that some waterworks prefer a slightly acid water (less than pH of 7) as it acts better in coagulating with lime only, provided the amount of acid, hardness, iron, and manganese can be kept within reasonable limits. Such water might be preferable to another of low alkalinity where an alkali and an acid-coagulating salt had to be applied for proper settling action.

The pH of natural waters is likely to range from 7.0 to 8.0, but organic acids may reduce the value to 5.0 and photosynthetic action may raise it to 8.0.[78]

In comparing contaminated water samples, therefore, the neutral point of 7.0 pH is not the correct line of division between a water that should be acceptable and a water that has been injured by acid contamination. The fact that the water in the reservoir had not dropped below a pH of 7.0 for several years until contamination developed is beside the point, as a water at a pH of 6.0 to 6.5 probably would have been about as satisfactory. Note that the water company was using aluminum sulphate as a coagulant for water with a pH of 7 or over, although lime which is much cheaper would have been equally satisfactory with a pH somewhat less than 7 (see Weston[148]).

ACCURACY OF pH DETERMINATIONS

Determinations of pH on acid-contaminated waters should be made electrometrically. Colorimetric determinations are difficult due to the varying color of the water, to the effect of acid in the water on the organic dyes in the color, to change in temperature of sample as taken and when test is made in laboratory, to use of artificial light in laboratory, to personal equation of observer, and to other factors listed in technical literature. Colorimetric methods at best even on normal waters are not closer than 0.2 pH and on acid-contaminated waters the error is frequently much greater.

Numerous sources of error in colorimetric determinations of pH even in waters between 6.6 and 8.0 have been outlined by McCrumb[79] and Dole.[80] Tengco[81] has reported errors from 0.2 to 0.4 pH in comparison of the glass electrode and colorimetric methods on waters ranging from 7.1 to 8.1. The glass electrode, however, may be used on highly colored water.[82] A comparison of pH values for several samples of acid-contaminated water, as determined by two chemists working independently, showed differences in colorimetric values ranging from 0.2 to 1.4 although the glass-electrode values were in close agreement.

Free hydrogen ions that cause change in the value of pH may come not only from free acid as such, but also from acid salts in mine drainage such as the sulphates of calcium, magnesium, iron, and aluminum. The iron and aluminum acid sulphates hydrolyze more readily than calcium and magnesium sulphates and therefore will depress the value of pH more.

Betz and Betz[83] emphasize that particularly in weakly buffered solutions the pH of the indicator may change the pH of the solution. Certain substances such as free chlorine and hypochlorite affect the color produced by the indicator and may cause erratic results. The pH of pure water at 50°F may be 7.2 but at 75°F it is about 7.0, a change showing the error that may be introduced between time of sampling and test.[70,81]

The tabulation below[84] gives an idea of how important errors in the correct determination of pH values become when the amount of acidity involved is a factor.

Change in pH	Approximate Equivalent Change in Acidity
0.1	1.25 Times
0.2	1.60
0.3	2.0
0.6	4.0
0.9	8.0
1.0	10.0

It follows from the preceding that an alkaline water may actually be highly contaminated with sulphate salts and that mine drainage of various degrees of contamination with sulphates of iron and aluminum cannot be safely compared on the basis of pH values.

FREE ACID

Free acid cannot be determined by usual methods with accuracy in the presence of the ferric and aluminum salts that are encountered in considerable quantity in acid-contaminated waters (see "Standard Methods"[85]). Free-acid determinations are possible by electrometric methods and the use of a chart showing grams of hydrogen per 1,000,000 ml (see Steel[86]). On semilog paper, the resulting plot is a straight line. Multiplying the grams of hydrogen corresponding to a given pH by 49 will give the ppm of free sulphuric acid. Similar multipliers are possible for other acids. Note that, based on normal weights, 1 g of hydrogen is equal to 49 g of sulphuric acid. Free sulphuric acid is usually the only important acid content in water contaminated by coal-mine drainage. Comparison of about 20 determinations of free acid by "Standard Methods" as reported by the water company and by colorimetric methods, showed errors in terms of the latter ranging from 15 to 3330 pct. Part of the error may arise from some instability with resulting progressive changes in the acid-contaminated waters, from errors in colorimetric values of pH, and from possible storage of samples overnight in a warm laboratory as compared to temperatures of probably about 50°F when samples were taken.

TOTAL SULPHATES AND TOTAL SOLIDS

Total sulphates is one of the most accurate chemical tests possible in comparing acid-contaminated waters. The method outlined in "Standard Methods,"[87] may be followed safely. Agreement within 2 or 3 pct may be reasonably expected of two chemists working independently. Silica should be eliminated if any appreciable quantity is present as otherwise the value for total sulphates will be too high.

Determinations of total solids should also show agreement within 2 or 3 pct if proper procedure is followed. Note that suspended solids should be included although the amount in most acid-contaminated waters may not be great.

PREFERRED ANALYTICAL BASIS

Instead of analyzing a contaminated water in ppm, it might lead to more informative results to determine the composition of a contaminated water on the basis of percentage of the total salinity.[61] The results will often be interesting as showing that an alkaline water has more contaminants in it as sulphates than an acid water (see Clarke[150]).

EXTENT OF CONTAMINATION

If determination is to be made of the extent of contamination of a reservoir from specified acid mine drainage, the amount must be expressed in pounds per day. It is not sufficient to take water samples and determine their soap hardness. Acid sulphates in ppm when combined with the volume of flow to get pounds per day or some other ratio is a fairer criterion. The discharge should be estimated from recording gauges giving the head at all times on each weir. Analyses of water samples at such frequent intervals as to coincide with marked changes in flow through the weirs will permit an approximate determination of contamination. Even such results must be corrected for the flow from hillsides directly into the reservoir below the points of sampling on the various streams.

TEMPORARY REDUCTION OF CONTAMINATION

Contamination during the interim between strip mining and contouring need not be important. Water should be removed by pumps from the pits at the coal face during stripping operations and thus there is little chance for contamination. What little acid water does escape from mine pools should be neutralized by the coal company before it reaches the reservoir of any water company. This step can be taken easily by bulldozing a small basin where the stream flow can be collected. A small spillway can be installed in an earth dam below the basin. Lime and even soda ash can be added by chemical feeders where the stream flow passes through the spillway. Below the spillway, another basin should be bulldozed so that the treated water will have time to settle out the precipitate formed by the chemicals.

The preceding treatment is too expensive on a large scale and leads to a large volume of sludge that is as difficult to handle as the original acid mine drainage.[11,18] It can be used to advantage, however, until the drainage has been reduced in quantity and strength by cutting off air and water from the contaminating sources.

Operators of strip mines should not overlook the possibility of diversion from one watershed to another of acid mine drainage where the contamination of a water supply is involved. Such diversion, however, does not solve the overall problem of stream pollution.

Factors in the Treatment of Acid-contaminated Raw Water for Domestic and Industrial Purposes

STANDARDS FOR SOFTENED WATER

Water contaminated with acid mine drainage can be made satisfactory for municipal and industrial use. It can be easily softened with lime and soda ash to a hardness of 150 ppm or less. A large portion of the country is using water in excess of 150 ppm hardness.[88]

Total solids of 1000 ppm are acceptable. American Water Works Association[89] points out that "in areas where water of any character is difficult to obtain, public supplies are used in which . . . the above limits (1000 ppm) are materially exceeded. In so far as the effects upon the constant user of such supplies are concerned no adverse conditions are known to ensue." (Compare U. S. Public Health Service,[90] Baylis,[91] and Collins, Lamar and Lohr.[88])

Sulphates as SO_4 are limited by the U. S. Public Health Service[90] and American Water Works Association[89] to 250 ppm, but Baylis[91] rates quality of water on sulphates ranging from 100 to 400 ppm. On the basis of comparative molecular weights, SO_4 at a maximum limit of 250 ppm is equivalent to sodium sulphate (Na_2SO_4) at 370 ppm. Many cities are using

supplies with sulphates ranging from 150 to 500 ppm.[88]

As to industrial use, limitations have been summarized by Lammers.[92] A municipal supply cannot meet all varied industrial demands so it is the custom of industry to put in such special water-treatment equipment as may be needed to make an available municipal supply satisfactory.

LIME AND SODA-ASH SOFTENING

The indicated treatment for acid-contaminated water is neutralization with lime and then the addition of soda ash to get the formation of the soluble sodium sulphate. Such softening can be done very well in a treating plant in which multiple recirculation of sludge, with incoming water and treating chemicals, results in instantaneous reaction for optimum formation of precipitate. Mechanical regulation of flow and direction of water through the plant will send the precipitate to its circulating zone and to its sludge concentrator and also will send the treated water into the clarified water chamber for discharge to the filters. An upflow type of clarifier and softener, therefore, is desirable in which provision is made for recirculating four or five volumes of sludge for each volume of raw water treated. Separation of solids is obtained by change in velocities and in direction of flow. The sludge blanket will not carry over to the filters.

Lime and soda-ash softening of acid-contaminated water has been shown repeatedly to be satisfactory. A few references supporting this statement are Trax,[93] Morgan and Young,[94] Keeler,[95] and Bouson.[96] Lime is the most efficient chemical known for reducing calcium hardness due to bicarbonate without adding any soluble salts to the water, for reaction with magnesium bicarbonates, and for neutralizing acidity.[97] Soda ash is the most economical means of removing the sulphate hardness that will be found in acid-contaminated water[97] (see Carpenter[151]).

EXCESSIVE LIME

Where lime is used in heavy dosages to change the pH of a water the intensity of an odor may be increased. This result can be offset by the use of chloramine which will also aid the action of any carbon used for removal of taste or odor.[98]

USE OF SODIUM ALUMINATE

Improved softening results are obtained in some cases with the use of a small amount of sodium aluminate (Clark and Price,[99] Black, Bardwell, and Graham,[100] Larson and Buswell,[101] and Hopkins[102]). Where the treated water is destined for boiler use, the addition of some sodium aluminate is particularly beneficial in lowering turbidity and increasing the weight of the sludge blanket.[77,100] The consumption of soda ash will be reduced in general about one-third pound for each pound of sodium aluminate. The liquid sodium aluminate will get better results than the dry form. One pound for each 10,000 to 20,000 gal daily will be needed at a cost of about 5½ cents per pound. Deducting the saving on soda ash, the net cost of sodium aluminate will be about 5 cents per pound (see Larson[149]).

USE OF SODIUM SILICATE

Sodium silicates can be used to increase rapidity of settling and extend the pH range over which coagulation gives good flocculation.[103]

USE OF ALUMINUM SILICATE

Aluminum silicates are helpful as an aid to coagulation. Their use is recommended to improve treatment and to reduce carry-over to the filters by increasing the weight of the precipitate and causing the formation of particle sizes that separate more readily from the clarified liquid. They are usually applied at the rate of about 30 lb per 1,000,000 gal and cost about 7½ cents per pound.

USE OF SEPTAPHOSPHATE

If trouble arises from the formation of aluminum hydroxide in the distribution system beyond the filters, an addition of septaphosphate ($Na_2P_7O_{22}$) will prevent such action. It is known as a stabilizer up to 200°F for any iron, calcium, magnesium, and alumina that may be in the water. About 8 lb will be needed per 1,000,000 gal daily at a cost of about 15 cents per pound.

pH VALUES FOR PRECIPITATION

The neutralization of acid in water with lime will bring the pH value up through 6.8 where any alumina should be precipitated. Further addition of lime will raise the pH through 8.2 to 8.4 where iron should be precipitated. The completion of precipitation of both iron and manganese will occur at a pH value of 9.2 to 9.4 (Bouson,[96] and Morgan and Young[94]).

REMOVAL OF ALUMINA

Acid-contaminated water will contain some alumina but there should be no difficulty in removing it when neutralizing the acid with lime (see previous paragraph). If successive additions of lime do not precipitate all the alumina, dolomitic limestone may be added. Its magnesium (about 40 pct) has an affinity for alumina and reverses the process of adding sodium aluminate to water to precipitate magnesium. Note that dolomitic limestone may be about 58 pct calcium as compared with a high-calcium limestone of about 95 pct calcium. The cost of dolomitic limestone and high-calcium lime as quicklime is about the same usually or $10 per ton. The use of dolomitic limestone in the small quantity needed to precipitate any alumina, therefore, will not increase materially the cost involved in neutralizing with lime.

And still further, if the precipitation of alumina continues to offer difficulties, an additional precipitation unit may be used in series with the first.

One manufacturing company in West Virginia, using acid-contaminated water, has had no difficulty in precipitating alumina with caustic soda.

ACID-CONTAMINATED WATER CAN BE HANDLED

The preceding data are supplied as the average water-treatment plant operator, faced with the problem of handling acid-contaminated water, will argue insurmountable difficulties that actually disappear when met with intelligent methods.

STRATIFICATION IN RESERVOIR

The poorest water in these contaminated reservoirs probably occurred in the summer near the bottom. Their maximum depth is about 25 ft. Acid mine water with its high mineral content will have a higher specific gravity than normal water and should sink to the bottom. Lack of dilution from rainfall and drawdown of reservoirs during

the summer should accentuate the bad effects of contamination. Reservoirs that were relatively long and narrow in comparison with their width probably caused short circuiting of contaminated water, from where most of it entered at the surface at opposite ends from the outlet gates, to those gates near the bottom. Relatively soft water, finally, was frequently wasted over the spillways at times of excessive rainfall and probably no attempt was made to draw off the contaminated water from the bottom of the reservoirs. The failure, therefore, to install variable-level, drawoff outlets probably made the contamination of the raw water in the treatment plant worse during summer and fall than it need to have been.

The preceding statements are supported by soap-hardness tests in August on 11 sets of samples from the two reservoirs. Of these sets 6 showed an increase in soap hardness in ppm, as between surface and bottom samples, ranging from 3 to 11 pct. No significant difference was found in the other 5 sets. A definite trend of 5 to 10 pct in increased soap hardness in the bottom samples was found as the point of sampling approached the outlet gates although surface samples showed no significant trend. For stratification in reservoirs, see Turneaure and Russell,[104] Flinn, Weston, and Bogert,[105] and Davis.[106] For increase in iron and manganese, see Wolman and Stegmaier,[107] Engineering News-Record,[108] Hopkins and McCall,[109] and Purcell.[110]

ESTIMATE OF DAMAGES TO WATER COMPANY

The company's first estimate of additional equipment needed to handle the acid-contaminated water was about $40,000 to $45,000 to cover a precipitator, chemical feeders, changes in its filter plant, and connecting piping. The precipitator was to have a capacity of 2,000,000 gal in 24 hr. At least $12,000, necessary to remodel the filter plant, represents obsolescence that should be borne by the water company. Even the entire amount could be regarded as a proper charge against the water company.

The water company's first estimate of additional operating expenses for lime, soda ash, recarbonation, power, and possible additional labor due to the acid-contaminated water was about $75 per 1,000,000 gal to remove 350 ppm of soap hardness. This estimate is reasonable as to amount if no allowance is made for the deterioration of the water company's supply due to causes over which the coal companies had no control. The assumption that the increase in hardness to 500 ppm, as compared with 150 ppm to which the water was to be softened, was due entirely to acid contamination is in error, as previously discussed.

An estimate of the increased operating cost, furthermore, should be based on the average soap hardness over a period of years and not on the peak soap hardness of a dry summer. The last complete year available is the dry year of 1944 when the yearly average was about 260 ppm. The average for the first eight months of 1945, including a dry summer, was about 390 ppm. Fall rains undoubtedly reduced the latter as the rainfall in September was about 10 in. or four times the average, and in November was about 4 in., or 1.6 times the average. Rainfall in October and December 1945 was about normal. The average soap hardness for 1945 undoubtedly was less than 350 ppm. The average soap hardness for the years 1944 and 1945 was probably about 300 ppm.

A previous tabulation has shown that prior to 1941, when the new stripping started, the average soap hardness of the reservoir water was 135 to 150 ppm with wet and dry weather respectively. The coal companies responsibility for contamination on a short-term basis, therefore, does not exceed 300 minus 150 or about 150 ppm.

But 150 ppm is about 43 pct of 350 ppm and 43 pct of $75 per 1,000,000 gal daily is about $32 daily. Based on 1,300,000 gal daily, the responsibility of the coal companies for increased softening costs does not exceed $15,200 yearly. Capitalized at 4 pct, this amount becomes $380,000 plus about $40,000 for new equipment minus about $10,000 for obsolescence in the filter plant or about $410,000 as compared with the payment of the equivalent of about $500,000 to the water company by the coal companies.

The preceding neglects the reduction in contamination that is possible with contouring. This reduction in the sealing of deep mines amounts to about 80 pct in a few years.[111] The residual of 20 pct of 150 ppm is 30 ppm or less than 9 pct of 350 ppm on which $75 per 1,000,000 gal is based. The charge to the coal companies eventually should not exceed about $6.50 per 1,000,000 gal daily or about $3100 yearly on 1,300,000 gal daily. Capitalized at 4 pct, this amount becomes about $80,000 plus $40,000 for equipment less $10,000 for obsolescence or about $110,000.

The proper payment for the coal companies lies between $110,000 and $410,000 and a payment of $300,000 would have been liberal as compared with the actual payment of about $500,000.

The preceding analysis, however, is still in error. It assumes the continued contamination by strip mining, improper control of drawoff in the reservoir, and drought conditions as during 1943, 1944, and 1945.

The conclusion is reasonable, therefore, that better presentation of the technical questions involved was desirable.

Factors in the Construction of a Water Plant

RESERVOIR CAPACITY

The capacity of any existing reservoirs should be checked to discover silting. Even in humid regions, the loss in capacity can be serious if the reservoir capacity is small compared with the annual runoff. In this case, the annual runoff could be about 1½ ft on 4.5 square miles or about 1,410,000,000 gal as compared with a total storage capacity said to be about 180,000,000 gal in two reservoirs (compare Davis[106]).

SPILLWAY CAPACITY

The spillway of an old reservoir should be checked for capacity against any state or Federal requirements. Old reservoirs often will be found to have inadequate spillways. The cost of increasing the spillway capacity to an adequate amount is a measure of obsolescence (compare Davis,[106] and Flinn, Weston, and Bogert[112]).

A runoff of 6 in. on 4.5 square miles is about 470,000,000 gal, or almost three times the total reservoir capacity, to be discharged in 24 hr—a condition that might easily arise.

FILTER OPERATION

The water company in this case claimed that the softening of the acid-contaminated water had increased the washing time on the filters to 2 hr each for four filters. This condition arose from improper prior treatment, coagulation, and settling of the contaminated water. A modern plant would not re-

quire more than 5 min to wash a filter at the rate of 15 gpm per square foot of area. It is probable that the cost of the excess water used in washing for 2 hr on each of four filters plus power and labor would justify the necessary investment of about $30,000 needed for an additional precipitator and connecting piping.

The filter plant in addition should be equipped with hydraulic valves, loss-of-head gauges, flow controllers, larger wash troughs, new underdrain system, rotary surface washers, and incidental modernizing equipment. Its cost at about $12,000 is a measure of the obsolescence in the treatment plant.

PIPE-LINE CAPACITY

A new supply for a water company usually means a new pipeline of considerable length from the new reservoir to the existing distribution system. The size of this line is frequently set by the rule-of-thumb assumption that a velocity of 2 to 5 fps in it will be satisfactory. This size should be checked to make sure the economic size has been selected based on cost of pumping, plus depreciation and interest charges on the necessary investment (compare Babbitt and Doland,[113] Davis,[114] Cameron,[115] and Reagan.[116]

Uses of Treated Acid-contaminated Water

LOCOMOTIVE BLOWDOWN

Blowdown losses cannot be estimated accurately for locomotives. An increase of 10 pct in water used due to blowdown means increased water and fuel costs that, it is true, can be estimated from data in Kent.[117] Increased blowdown means cleaner tubes, lower maintenance costs, and greater efficiency in burning all fuel so that any fuel lost in blowdown may be offset.

Curtiss[118] has made an intensive study of the value of water treatment to the New York Central Railroad and concludes it is at least 13 cents per pound of incrusting solids. He says that continuous automatic blowdown costs about $18 per month per locomotive but the saving in washout expense is about $36 per month per locomotive.

Locomotives are frequently equipped with continuous blowdown plus manually-operated valve, steam separators, and electric foam-collapsing systems. Blowdown of 10 pct on the main line of a large eastern railroad is desirable with 15 to 20 pct for a smaller road (Hislop,[119] and Williams[120]).

Limit of Total Solids in Feedwater

The amount of blowdown necessary is determined by the limit possible on total solids. If this amount is 1000 ppm and the feedwater has 3000 ppm, then the required blowdown is 33 pct.

The desirable limit for total solids is 3500 ppm for 0 to 300 lb pressure (compare Cassidy,[117] The American Boiler Manufacturers' Association and Affiliated Industries' Fair Practice Committee,[121] and Lammers[92]).

Treatment

Excellent results have been obtained in reducing locomotive-boiler blowdown with organic colloids in feedwater treatment. Concentration of solids from 3400 to 14,000 ppm has been reported with reduction in blowdown from 10 to 1 pct and extension of washing out period from 4 to 10 days (de Frank[122]).

Specially prepared amines and amides are organics that are definite antifoam compounds. They get better results than castor oil. Dissolved solids of 10,000 or more are known and 5000 ppm are common in boiler operation.[120]

Foaming is not usually encountered until the concentration reaches 2100 to 3400 ppm. Castor oil in tannin extract is said to control most cases of foaming. An increase in the concentration of dissolved solids by 75 to 100 pct is claimed.[123] Various types of tannins are also available.[124]

Most types of oil and organic matter will increase foaming.[120] Numerous compounds, however, are available such as cholesterol oils from dried sewage, emulsified castor oil, and various vegetable oils, all of which are more or less effective in control of foam.[125]

No conclusive proof exists that suspended matter is the cause of foaming.[120,125] But certainly filtered water is an advantage as eliminating one possible cause.

One railroad has used sodium aluminate plus organic materials in increasing concentration to 10,000 to 30,000 ppm and reducing blowdown to 2 to 6 pct.[126]

No standard treatment for feedwater is possible but each problem can be solved by the railroad as it develops.[120,127] Terrific concentrations of solids are possible under some conditions without foaming.[120]

The preceding data lead to the conclusion that a filtered water with total solids under the worst conditions not exceeding 1000 ppm, mainly as the soluble salt sodium sulphate, can be handled in locomotive boilers with satisfaction.

Another bug-a-boo that has been raised over the use of an acid-contaminated water which has been softened by lime and soda ash is the content of soluble sodium sulphate.

MEDICAL USE OF SODIUM SULPHATE

Sodium sulphate is also known as Glauber's salt. The latter is sold for laxative purposes and has the chemical formula $Na_2SO_4 \cdot 10\ H_2O$ with molecular weight of about 322. Sodium sulphate (Na_2SO_4) as it occurs in water is without the water of crystallization and has the molecular weight of about 142. The sulphate radicle (SO_4) has the molecular weight of 96.

Merck's Index[128] gives the medicinal dose for Glauber's salt for laxative purposes as 15 g. Fitch[129] recommends for combined salts a dose of 12 g. Assuming a dose of 15 g, the anhydrous sodium sulphate required in a solution of water will be in proportion to the molecular weight or 6.6 g. When sodium sulphate is 1000 ppm, the amount of water necessary to get 6.6 g will be 6600 g. But 1 lb is 453.6 g, and therefore a person would have to drink 14.5 lb of water or 1.75 gal in a very short period.

Fitch[129] says the minimum dose recommended is 1.5 g taken in one large draught of water or say one drinking glass full, as otherwise it will not produce laxative action. But 1.75 gal is equivalent to about 28 drinking glasses. Fitch recommends, therefore, for laxative action a dose of 42 g in 1.75 gal of water. This amount is more than six times the dose obtained when sodium sulphate is 1000 ppm.

In comparison with the large amounts of water mentioned above, note should be made of what a man needs as fluid daily. Sollman[130] claims that from 0.8 to 2.0 liters daily are required and that drinking one liter at each meal is a large amount. McLester[131] places the proper amount at one to two liters daily for the healthy man and says the average adult should take 1.5 to 2 liters daily. In summer and especially in warm climates, these amounts should be increased. Note that one U. S. gallon is equal to about 3.8 liters.

SODIUM SULPHATE IN WATER

Experience with sodium sulphate in drinking water has shown that it is not so objectionable as is often assumed up to concentrations of 1000 ppm (Ellis and Meinzer,[132] Mendenhall, Dole, and Stabler,[133] Suckling,[71] Fitch[129]).

Dixey[134] says that water with 2500 ppm of dissolved salts may be used for many days without serious discomfort, but that water with 3300 to 4000 ppm is used. The immediate consequence of drinking water too high in mineral content is usually diarrhea, but people gradually acquire immunity. Of the various dissolved salts in water, the alkaline sulphates are the least injurious.

Medical testimony also does not support the claim that sodium sulphate is particularly objectionable in water (compare Cushny,[135] and Sollman.[136])

There is no scientific evidence that waters high in mineral content cause kidney trouble or gallstones. Any occasional mild laxative action that such water might cause is an idiosyncracy that should pass quickly with longer use of the water. The taste might be somewhat brackish to the unaccustomed individual.

For some contrary opinions see Boaz,[137] Cohen,[138] Mason and Buswell,[139] Dole,[140] McLester,[131] and Trax.[93]

An acid-contaminated water that has been softened to 150 ppm or less, filtered, and does not contain more than 1000 ppm of dissolved salts as sodium sulphate, in the light of the preceding, cannot be considered an objectionable water.

Physiological Aspects

There is no evidence that hard waters even up to 500 ppm[141] are more or less wholesome than soft waters. Harmful physiological effects from iron, manganese, and aluminum[142] need not be expected up to high concentrations. Amounts retained in filtered water after coagulation are so small that only the most sensitive micromechanical methods will reveal such minerals. No reliable published paper[143] is available that definitely indicates hard water is physiologically harmful. Taste is more important than mineral-salt content but Negus[147] claims that taste is not necessarily harmful. The acid-contaminated water in this case at its extreme of about 500 ppm could be easily softened to 150 ppm or less. The argument, therefore, that a new water supply was necessary physiologically because of hardness falls to the ground.

LIMITS OF SOLIDS FOR CATTLE

Dissolved salts up to 3000 ppm are accepted in Victoria, Australia[144] as safe for working horses, dairy cattle, and pigs. Stock will habitually drink water twice as high in mineralized content[132] as man will accept. Stock has been known to drink water with a sulphate radicle as high as 4800 to 5600 ppm. An acid-contaminated water, therefore, that has been softened to 150 ppm and has 1000 ppm of sodium sulphate should be considered excellent for cattle.

Conclusion

Methods that may be used to reduce acid-mine drainage from strip coal mines include contouring to increase surface run off and such sealing methods as cement grouting, use of sheet piling, chemical impregnation, use of tar blankets, and applications of bentonite. The success of each is based on the theory that any reduction of air and water or either in contact with iron sulphide in the coal or shale will reduce the amount of acid formed.

Meteorological factors such as rainfall, temperature, barometric pressure, wind, and diffusion are of little consequence in connection with contouring in their effect upon the amount of air in contact with iron sulphide. An exchange of air is one of the factors necessary in continuous acid formation.

The hardness of water is increased by dry weather. A swamp will often reduce such hardness by the diluting action of its stored water. An alkaline water can be as objectionable in its sulphate content as an acid water. Acid contamination, therefore, cannot be considered without also studying weather and topographical conditions.

Several of the usual chemical methods for the analysis of water are not satisfactory for acid mine drainage. This statement applies particularly to values of pH, soap hardness, and free-acid determinations. Comparisons of acid mine drainage based on such values will be misleading

Acid mine drainage can be treated to make it satisfactory for domestic and commercial use. The first step is to reduce the quantity of such drainage. The next step is so to draw off from any reservoir the contaminated water that the contamination is reduced. Then various chemicals may be used in a water-softening, filtration plant. Many of these chemicals are not ordinarily used in water-treatment practice.

Damages to a water company from acid-contaminated water should not be assessed without considering losses in value of the water-company's facilities arising from silting of reservoirs, inadequate spillway capacity, excessive pipeline capacity, and obsolescence in any filter plant.

Treated acid-contaminated water can be used satisfactorily for locomotive water supply and domestic purposes with far higher concentration of salts than is commonly supposed. Extravagant standards are sometimes set for a water supply that are not justified by general water-supply conditions or by the competing requirement from industrial growth. The important point is to arrive at such a reasonable balance among conflicting demands as will permit adequate service both to industry and municipalities.

The contouring policy of the Sunnyhill Coal Co. attempted the following: When its operations were completed, the mined area was left looking as attractive as or even more attractive than the natural country when stripping was started. An attempt was made to meet the critics of stripping operations who strongly object to ugly spoil banks and contaminated water (compare Sappenfield[145]). With the low-cost methods now possible with large, modern machinery such as drag lines, stripping shovels, coal-loading shovels, and other motorized equipment such as large trucks, the cost of contouring was not excessive in the case of the Sunnyhill Coal Co.

Pennsylvania has an economic asset in its thin beds of coal that can be mined economically only by stripping. As Toenges and Anderson[146] point out, deep mining recovers only 50 to 60 pct of the coal available whereas strip mining recovers 75 to 95 pct of the coal that otherwise would be a total economic loss.

The methods of the Sunnyhill Coal Co. are worthy of examination where conflict arises between public demands and the requirements of stripping operations.

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Correlation of the Performance Characteristics of Domestic Stoker Coals with Their Chemical and Petrographic Composition

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Introduction

One of the most urgent needs in the field of coal combustion is the ability to predict the performance of a coal from knowledge gained from small-scale tests. Numerous types of analyses and tests are conducted on coal, including the proximate and ultimate analyses, heating value, varieties of sulphur, ash analysis, ash fusion temperatures, free swelling index, petrographic analysis, and Gieseler plasticity. The Illinois Geological Survey has made an intensive study of the value of these tests for judging the performance characteristics of the coal as actually utilized. To date, three reports[1,2,3] have been issued, and another is being prepared. The present paper briefly summarizes some of the results found in the studies.

Equipment

The equipment used for the tests is described in some detail in the above mentioned publications. Briefly, it consisted of a domestic stoker, cast-iron boiler, and water-cooled heat exchanger, which were operated as a forced circulation hot-water system. The entire unit was mounted on scales.

Instruments for recording the performance of the coal included a hot-water meter to indicate the quantity of water flowing through the boiler; a two-pen, mercury-actuated thermometer to record the temperatures of the water entering and leaving the boiler; a chemical-type meter to record the percentage of CO_2 in the stack gases; a pressure gauge to record the static pressure in the stoker-air duct; and a multipoint potentiometer to record the temperatures in the stack and room, and the opacity of the stack gases. A 16 mm motion picture camera was available for taking pictures of either the fuel bed or the scale dial.

White Sulphur Springs Meeting, November 1949.

Paper presented at Joint Fuels Conference of Coal Division AIME and Fuels Division ASME.

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[1] References are at the end of the paper.

Procedure

The tests described in this report can be conveniently divided into three phases. For the first phase, a load of 5 or 6 tons of unwashed coal (usually screenings) was obtained from each of fourteen Illinois mines. These coals were all crushed and screened to a size of ½ in. (square hole) by 8 mesh. Combustion tests were made on a part of this raw, screened coal. Another portion was passed over a concentrating table to reduce the ash to what was considered to be commercially reasonable, and combustion tests were made on this upgraded coal. The ash content of the remaining coal was reduced to a still lower figure by passing it over the washing table with greater reject, or by standard float-sink procedure. This cleaned fraction was then tested in the stoker-boiler unit. Thus three coals with varying ash content were tested from each mine.

For the second phase reported here, coals were obtained from four Arkansas mines and tested as received and after passing over the concentrating table with a normal reject. One Illinois coal was tested in the "washed" condition only.

The third phase included tests on commercially prepared stoker coals

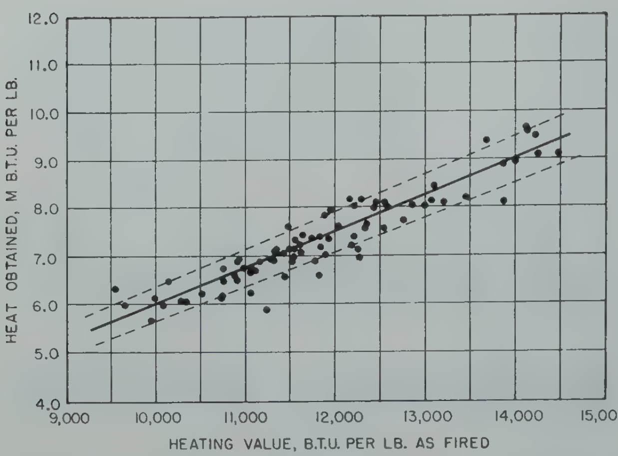

FIG 1—Relationship of heat obtained per pound of coal to heating value, on the as-fired basis.

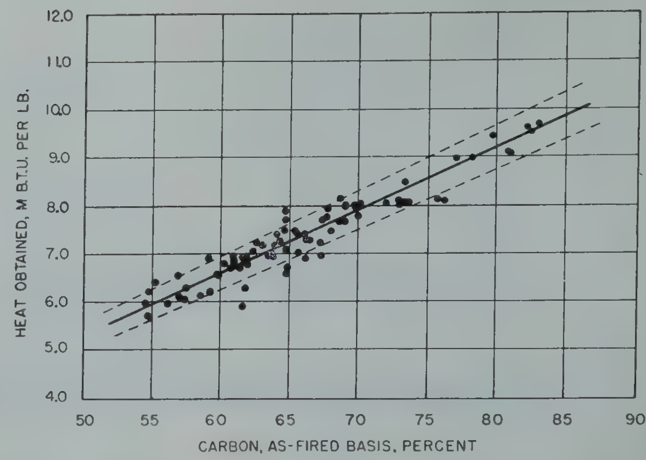

FIG 2—Relationship of heat obtained per pound of coal to percentage of carbon, on the as-fired basis.

from Illinois and several other states and from numerous coal seams. If feasible, they were obtained directly from the mine; if not, they were obtained from a retail coal dealer. There is a possibility of improper identification of the coals obtained in this latter manner, but all reasonable precautions were taken. Table 1 gives information about the source of coals tested in the second and third phase. A previous report[2] gives similar information about the first phase.

The sample obtained is not considered as representative of the seam, or even the unnamed mine. It was obviously impractical, and of no particular importance, to obtain a representative sample for this study. The coals were obtained from a large number of sources in order to have coals of widely varying characteristics.

Table 1 . . . Source of Coals for Second and Third Phase Tests

State	County	Seam
Illinois	Douglas	No. 7
Illinois	Grundy	No. 2
Illinois	Franklin	No. 6
Illinois	Fulton	No. 5
Illinois	Knox	No. 6
Illinois	Grundy	No. 7
Illinois	Christian	No. 6
Illinois	Saline	No. 5
Illinois	Saline	No. 6
Illinois	Randolph	No. 6
Illinois	Jackson	No. 6
Illinois	Bureau	No. 6
Illinois	St. Clair	No. 6
Arkansas	Sebastion	Hartshorne (2 samples)
Arkansas	Johnson	Hartshorne (Spadra)
Arkansas	Logan	Paris
Indiana	Vigo	No. 3
Indiana	Vigo	No. 4
Indiana	Gibson	No. 5
Indiana	Greene	No. 6
Indiana	Greene	No. 7
Indiana	Fountain	Minshall
Indiana	Clay	Brazil block
Kentucky	Harlan	Darby No. 5
Kentucky	Perry	Hazard No. 4
Kentucky	Hopkins	No. 11
Kentucky	Hopkins	No. 6
Kentucky	Letcher	Elkhorn
Missouri	Randolph	Bevier
Missouri	Macon	Mulky
West Virginia	Marion	Pittsburgh
West Virginia	Wyoming	Beckley
West Virginia	McDowell	Pocahontas No. 4
West Virginia	Webster	Sewell
Wyoming	Sheridan	Monarch

Considerable care was used in obtaining a representative sample of the coals as burned, for chemical and petrographic analyses. Standard or proposed methods of the ASTM were used wherever they were applicable.

The petrographic composition was obtained only for the Illinois coals in the first phase. Those listed in this report were made by microscopic examination of closely sized fractions. The analyst considered all vitrain

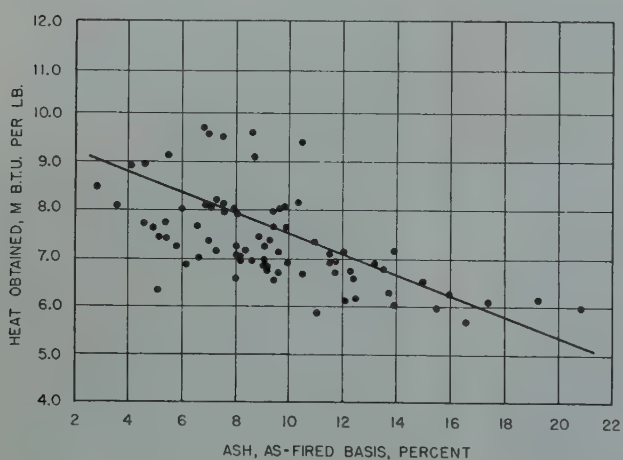

FIG 3—Relationship of heat obtained per pound of coal to percentage of ash, on the as-fired basis.

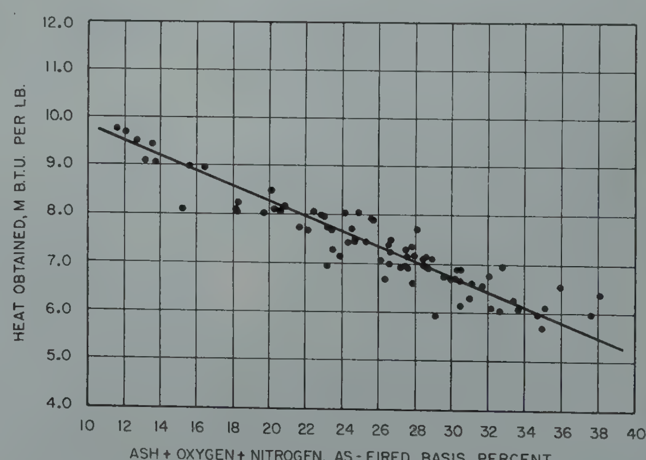

FIG 4—Relationship of heat obtained per pound of coal to the sum of the percentages of ash, oxygen, and nitrogen, on the as-fired basis.

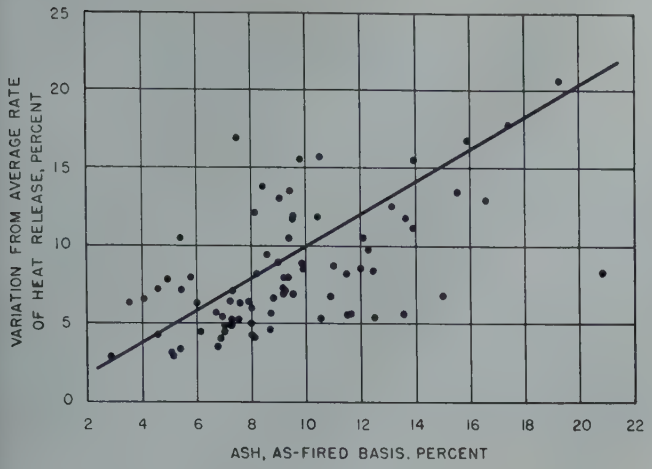

FIG 5—Relationship of the uniformity of rate of heat release to percentage of ash, on the as-fired basis.

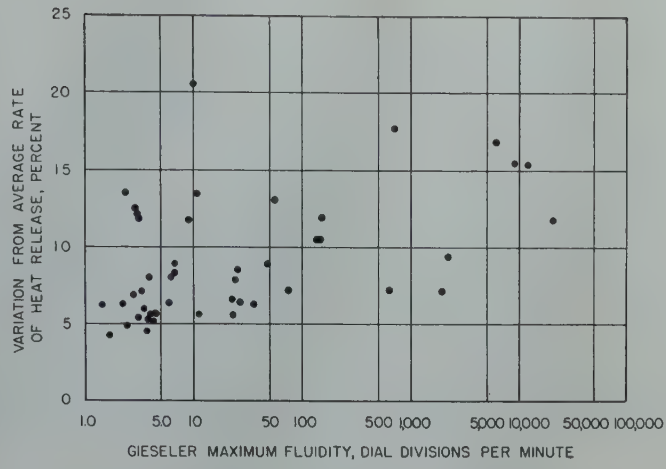

FIG 6—Relationship of the uniformity of rate of heat release to the maximum fluidity, in dial divisions per minute, as determined by the Gieseler test.

bands less than 1/10 in. thick as part of clarain.

The combustion testing schedule included five stoker-operation rates varying from hold-fire to continuous. About 300 lb of coal were burned during each of the operation rates, except hold-fire. No attention was given to the fire during each of these tests with a fixed operation rate.

Combustion Rating Criteria

There are many factors that govern the suitability of coals for domestic stokers. Included are: (1) amount of heat obtained per dollar, (2) attention required by heating plant, (3) ability to maintain the desired temperature in the home, (4) smoke emitted, (5) ability to maintain fire at low rates of operation, (6) cleanliness, (7) the appearance of the fuel bed and fire, (8) the odors given off by clinkers during their removal, (9) quietness of operation, and (10) appearance of the coal. These factors vary in relative importance, depending upon the heating system being used and also upon personal preferences of the operator.

Objective measures of all of these factors would be highly desirable. Unfortunately only the cost of heat can be determined by generally recognized standard tests and this only under arbitrarily fixed conditions. While the cost of heat may be of major importance with large stoker-fired heating plants, it is considered to be of minor importance to many domestic stoker owners.

Some objective criteria of the first five of the factors listed above were devised and determined for all the coals tested. The chief criterion of the attention required is considered to be the quantity of ash, which can be obtained from the proximate analysis. The percentage of ash per million Btu is a more precise measure. Other criteria used were the density and friability of the clinker, and the relative amount of ash removed in the form of clinkers. A subjective clinker rating was also made.

The ability of a heating plant to maintain the desired temperature in a house will depend upon many other factors besides the coal being burned. However, certain performance characteristics of the coal are thought to be influential and were determined for all coals tested. These were the uniformity of heat release, ratio of the minimum and average rates of heat release with

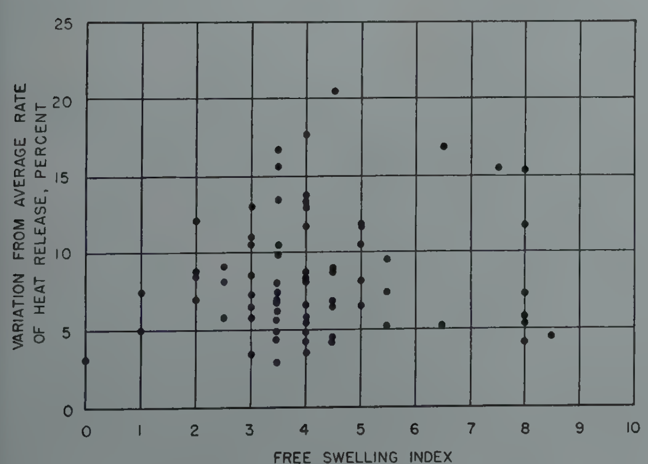

FIG 7—Relationship of the uniformity of rate of heat release to the free swelling index.

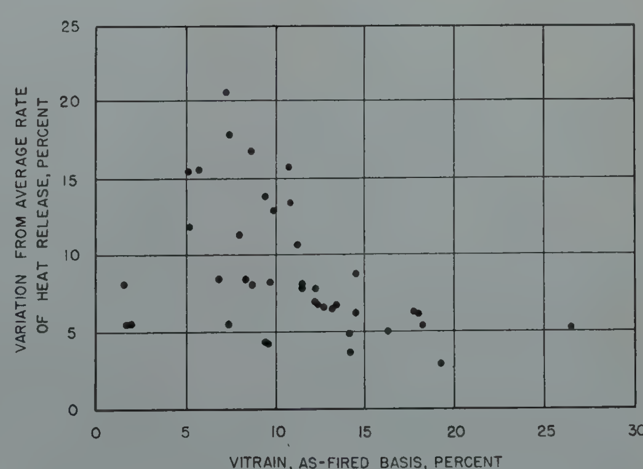

FIG 8—Relationship of the uniformity of rate of heat release to percentage of vitrain.

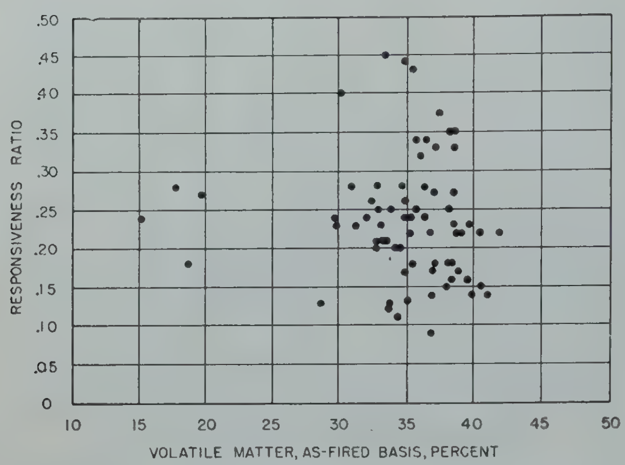

FIG 9—Relationship of the responsiveness of the fire after a prolonged hold-fire period to percentage of volatile matter on the as-fired basis.

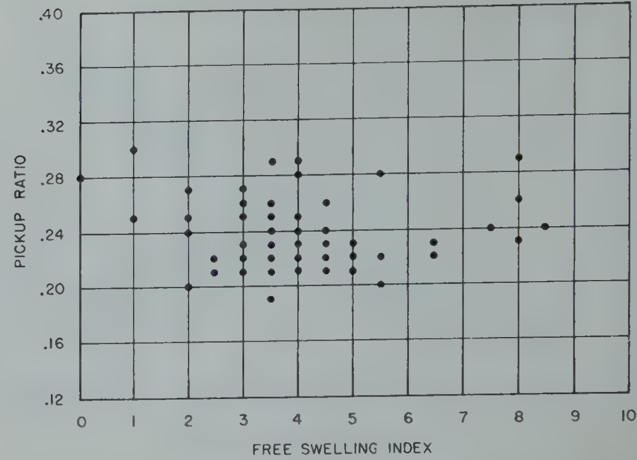

FIG 10—Relationship of the pickup rate after a 45-minute "off" period to the free swelling index.

continuous stoker operation, responsiveness of the fire after a prolonged hold-fire period, the "pickup" after a relatively short "off" period, and the rate of heat output after the stoker shut-off (called "overrun" in this report).

Although no discussion of smoke and hold-fire ability is given in this report, objective data were obtained. Motion pictures were taken of the fuel bed at fixed intervals, and a considerable quantity of colored film showing this action is available for study.

Results

HEAT OBTAINED

The amount of heat obtained from each coal tested was very nearly directly proportional to the heating value of the coal, on the as-fired basis (Fig 1). The solid line appears to be the best single line to represent the points shown. All points falling within the dotted lines are within 5 pct of the value indicated by the solid line.

The heat obtained from the coals tested was also nearly directly proportional to the percentage of carbon as given in the ultimate analysis (Fig 2). Most of the points are within 5 pct of the indicated line.

A fair correlation between the ash and heat obtained was found (Fig 3). This relationship is of particular interest because the percentage of ash is also a criterion of the attention required. Ash is not the sole diluent in coal, and as expected, a better correlation with the sum of noncombustibles was shown (Fig 4).

Numerous other items, and combination of items, gave good correlation with the heat obtained. However, none appear to be as useful as the relationships mentioned, particularly the one with heating value.

Uniformity of Rate of Heat Release

None of the items included in the chemical or petrographic analyses gave a good correlation with the uniformity of rate of heat release. One of the better correlations appeared to be with the percentage of ash (Fig 5). The performance of several coals failed to follow the general trend of increasing variability in rate of heat release with an increase in percentage of ash. One example of nonconformity was the Pittsburgh seam coal from West Virginia which had about 17 pct variation, although its ash content was only 7.5 pct. An unprepared coal from Wabash

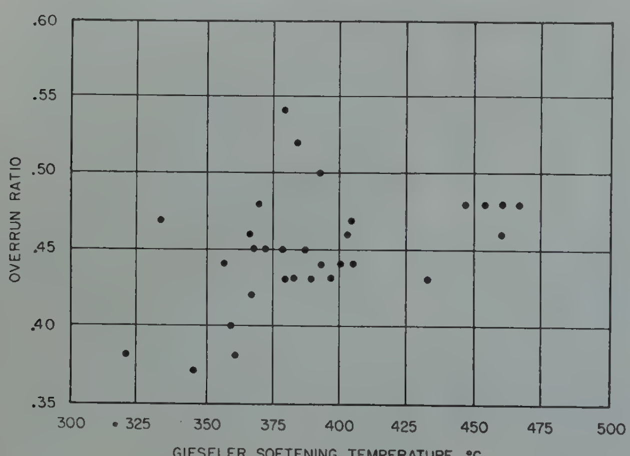

FIG 11—Relationship of the overrun rate after a 15-minute "on" period to the Gieseler softening temperature.

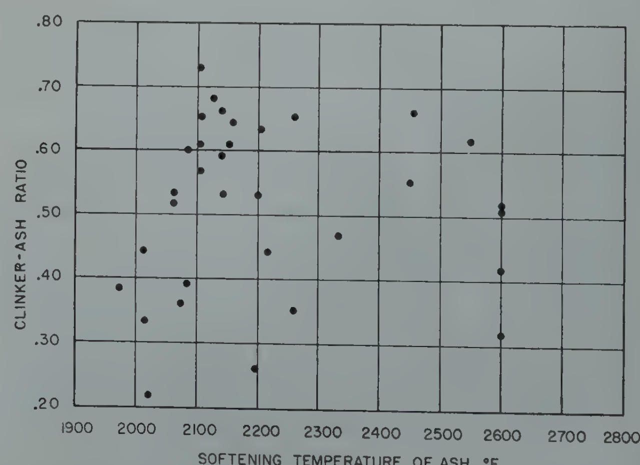

FIG 12—Relationship of the clinker-ash ratio to the softening temperature of the ash.

County, Ill., burned with only 8.3 pct variation, yet its ash content was 20.8 pct.

Fig 6 shows the correlation of the uniformity of rate of heat release and the maximum fluidity, in dial divisions per minute, as determined by the Gieseler test. Although the degree of correlation is probably poorer than that exhibited with ash, the Pittsburgh seam coal (abscissa of 6000) conforms to the general relationship shown. The Wabash County (Ill.) coal is not shown because no plastic properties were indicated by the Gieseler procedure.

The coking characteristics of coals are usually thought to exert considerable influence upon the uniformity of combustion in domestic stokers. One of the best indicators of coking tendency is thought to be the free swelling index. However, Fig 7 shows that the correlation is very low. Five coals with swelling indexes of 8 or greater burned quite uniformly.

The banded ingredients were not determined for all of the coals burned because of lack of suitably trained analysts. Fig 8 shows the relationship between the uniformity of rate of heat release and vitrain for all coals analyzed. The general tendency seems to be for more uniform combustion with the higher vitrain coals. This may be a reflection of ash content, because the coals with higher vitrain usually had lower percentages of ash.

As stated, the percentage of vitrain reported did not include any bands less than 0.1 in. in thickness. The coals with "bright" clarain might have considerably more vitrain than reported, if no limit were placed upon the thickness of the bands. The correlations with the banded ingredients are therefore considered to be incomplete. It is hoped that further investigations can be made.

Responsiveness of Fire to Heat Demands

No useful correlation was found between any of the chemical or petrographic tests and the responsiveness of the fire after a prolonged hold-fire period. The relationship of responsiveness and volatile matter is shown in Fig 9. The widely scattered points were typical of all plots made.

The correlation of the responsiveness of the fire after a 45-min off period (called pickup) with any chemical or petrographic item was likewise poor.

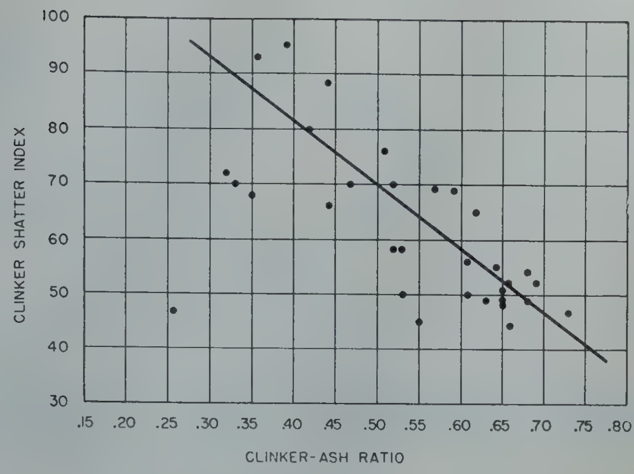

FIG 13—Relationship of the clinker shatter index to the clinker-ash ratio.

Fig 10 shows a typical example.

The rate of heat release after the stoker shut off did not exhibit a useful correlation with any of the chemical or petrographic tests made. A slight tendency for an increase in "overrun" to accompany an increase in the Gieseler softening temperature is shown in Fig 11.

Clinker Characteristics

One of the primary requisites of coal for a "clinkering" type stoker is the ability to remove the ash in the form of clinker. This characteristic was indicated as the ratio of the weight of clinker removed during the tests and the total weight of ash formed. The ratio was always less than unity because the tests were started with no ash in the combustion chamber. A bed of loose ash had to be accumulated before clinker was formed. Less than unit values do not mean that all the ash could not be removed in the form of clinker with normal home operation.

Fig 12 shows that the correlation of the softening temperature of the ash and the clinker-ash ratio is very poor. The clinker-ash ratios for several coals in the 2000°F range were lower than some in the 2600°F range.

There appears to be a slight correlation between the clinker-ash ratio and the clinker-shatter index* (Fig 13).

Conclusions

The heat obtained from the coals tested exhibited a good correlation with several items given by chemical analyses. The most useful relationship was with the heating value, on the as-fired basis.

No single item, or combination of items given in chemical or petrographic analyses was found that gave good correlations with any of the other measured combustion characteristics. The percentage of ash, the fluidity of the coal as determined by the Gieseler test, and the percentage of vitrain exhibited a fair correlation with the uniformity of combustion.

The clinker-ash ratio appeared to have a fair correlation with the shattering characteristics of the clinker.

Insufficient space is available to show additional graphs or to discuss the lack of correlation exhibited by more than one hundred combinations of chemical and combustion characteristics that were studied. Little or no useful correlation was exhibited by any except those mentioned in this report.

* The percentage of clinker by weight that broke into pieces weighing less than ¼ lb after two drops from a height of 6 ft.

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Ready-made Heat from Coal

BY D. W. LOUCKS*

There is plenty of evidence to indicate that at least one of man's chief interests in life is to make himself as comfortable as possible. If you doubt this, just watch the fellow next to you for the next half hour trying to find the most comfortable position that a hard chair has to offer. Comfort, however, does not always mean an easy chair. To some, it may mean a wealth of money; to another, freedom from worry. But to most of us, it means first of all a comfortable atmosphere in which to live, and to a great many of us it probably also means freedom from that annoying task of firing the furnace.

Today more than ever before, automatic heat is one improvement that is placed high on everyone's list. Perhaps this is because automatic heating is becoming relatively cheaper. Perhaps it is because of a good publicity campaign on the part of the oil and gas men or maybe it is just that we are getting lazier day by day. At any rate, almost every issue of *Better Homes and Gardens*, *House Beautiful*, or your other favorite home magazine carries an article extolling the virtues of this or that automatic heating system.

If I were to ask you to name the first thing that came to your mind when I said automatic heat, you would probably say either gas furnace or oil burner. Or if you had just been studying heating systems, you might possibly say heat pump. But chances are you would not mention anything about coal, and yet coal is the most common source of the greatest automatic heat of them all. I say this because coal is the fuel used almost universally by the district heating industry in producing and delivering to certain heavily populated areas heat ready to use at the touch of a valve or the click of a thermostat. Although the industry is over a half century old, it has not experienced the widespread development of other utility industries because of certain limitations which I believe you will realize from the next few minutes discussion.

District Heating Operations

We may define district heating as any operation where two or more buildings are heated from a central heating plant. The method of heat transfer may be hot water or in some cases warm air, but generally the medium of heat transfer is steam. So universally is steam used that the industry is frequently referred to as the district steam industry.

The Allegheny County Steam Heating Co. which operates the district heating system in downtown Pittsburgh is a subsidiary of the Duquesne Light Co. Although organized in 1912 primarily as a means of securing the electric load of downtown buildings, the service has now become so valuable and so popular that it is no longer considered a necessary adjunct to the electric business but rather a separate business standing on its own feet.

Fig 1 shows the layout of the plants and distribution system of downtown Pittsburgh. Two generating plants, one known as the Stanwix and the other as Twelfth Street, supply the area. Each has two boilers with capacity totaling 1,350,000 lb per hour. The Stanwix Plant is supplied coal by truck. The coal is pulverized at the plant and burned as powdered fuel. Coal is supplied to the Twelfth Street Plant also by truck but the boilers are stoker fired.

Over 1½ miles of tunnel house a portion of our main lines, but it requires over twelve miles of pipeline, ranging in size from 32 down to 1 in. in diameter, to supply all our customers. The distribution system consists of two systems in a sense, one high and one low pressure with certain interconnections between the two. Our high pressure system supplies steam up to 125 lb to some but not all customers, while the low pressure system operates in the range of 10 to 20 psi. Note that the two plants are tied together through large steam mains and that the system to some extent is a loop system, making it possible to have a portion of the line shut down without interrupting service to any customer.

Fig 2 conveys a picture of the extent to which steam service is used in the downtown triangle. The black area indicates the buildings which now use district steam. The dotted area indi-

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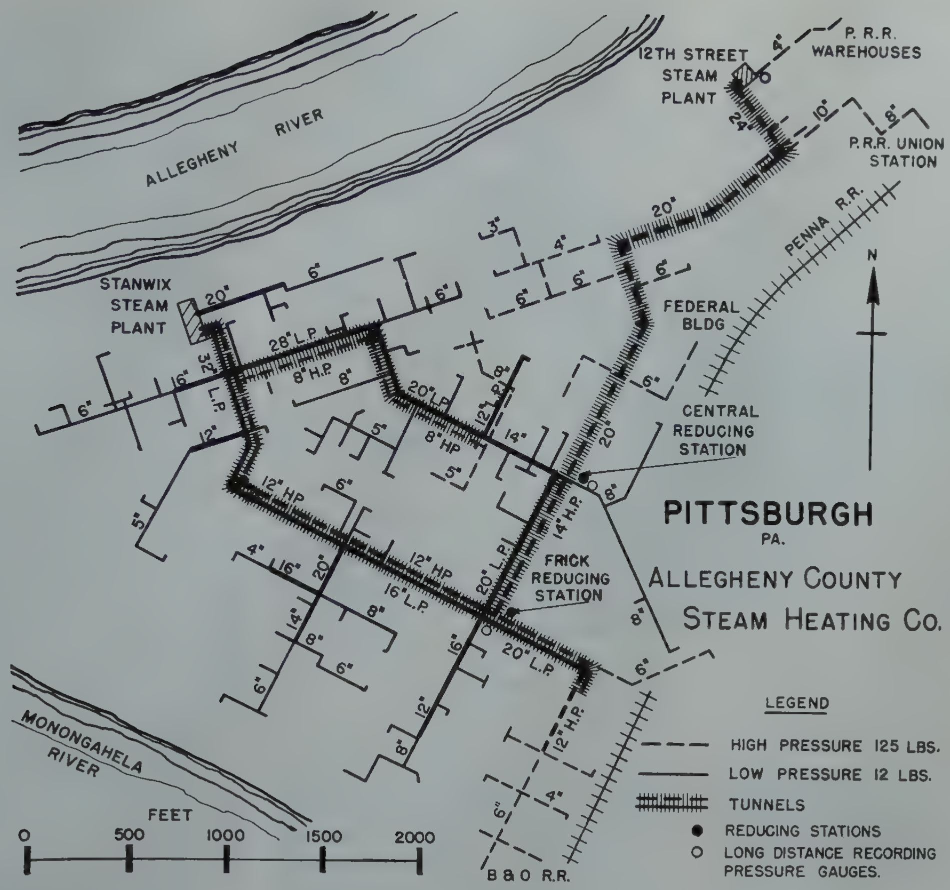

FIG 1—Layout of plants and distribution of system in downtown Pittsburgh, Pa.

cates the vacant land or parking lots, and the white area indicates buildings not yet using service. Note that practically all of the major buildings in the area are supplied, including such as the Oliver Building, Kaufmann's Store, Gulf, Koppers, Grant Buildings, the U. S. Post Office, the City County Building, and many of the smaller buildings and stores. Certain sections are devoid of users. These are areas which have not yet been developed, either because the amount of business to be secured is not sufficient to justify the cost of line extensions or because expansion in other districts has been more promising, and we have not yet developed the other territories.

For many years the downtown has had very little new building, and our growth has developed from existing buildings operating their own plants. Recent talk of new buildings in the downtown area lends encouragement to us as well as many others, and we are confident that such new building development will result in further expansion and growth of the district heating business in Pittsburgh.

Growth of District Heating in Pittsburgh

The Company started in 1912 with six customers, and it has grown slowly but steadily ever since. Last year, we sold approximately 1,450,000,000 lb of steam to 465 customers for which we received revenue amounting to about $1,350,000. To supply that many customers with that much steam requires an investment here in Pittsburgh of over $8,000,000 of which approximately half is in boiler plants and the other half in distribution facilities. Last winter the hourly output reached 850,000 lb of steam which is equivalent to converting 425 tons of water to steam in 1 hr. That is a considerable quantity of steam but when compared to our total generating capacity of 1,350,000 lb per hour, it is easy to see that we still have plenty of spare capacity.

A careful analysis of the foregoing figures will reveal some rather interesting features of the district heating business. For example, a brief calculation will show that the average amount paid us per customer is $3000 per year, and the average net rate per thousand pounds of steam sold is between 93 and 94 cents. Further analysis will indicate that the annual sales of 1,450,000,000 lb of steam is only 20 pct as much steam as the peak load facilities of 850,000 lb per hour are capable of producing. When this 20 pct load factor is compared to one of 65 to 75 pct for the electric utility industry, it is easy to pick out one of the factors that so far has been a handicap to us. We are, as you know, in a seasonal business, largely building heating. What we need is a large summer business to fill in the valley. Several possibilities are constantly being studied by the industry. One is the development of a year-round manufacturing process load. Another is the development of the use of district steam for air conditioning. This latter appears to offer considerable possibilities especially for commercial areas such as we supply where there is little hope of developing much industrial load.

The figures just quoted also reveal that nearly $6 of invested capital is

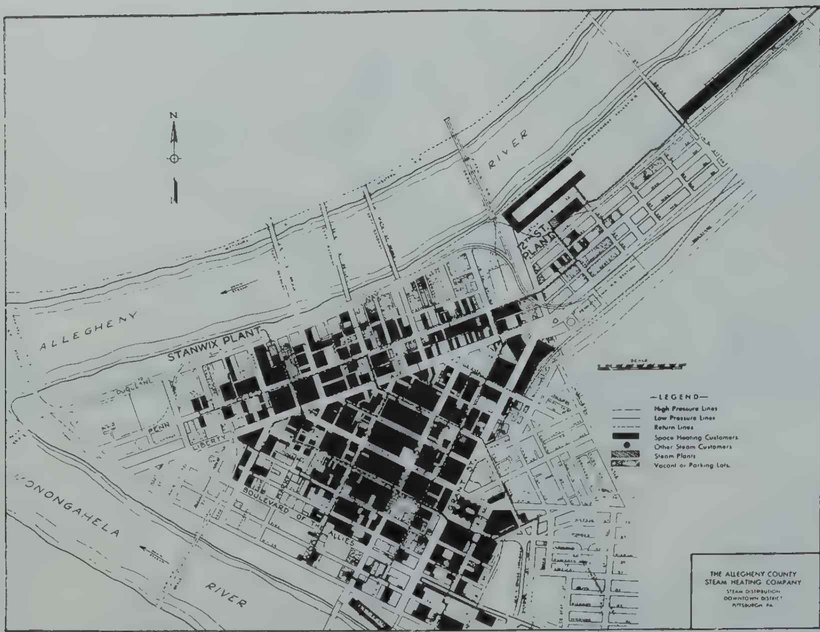

FIG 2—Steam distribution in downtown district of Pittsburgh, Pa.

required to produce one dollar annual revenue. This, of course, varies in different cities but these figures reveal two other significant features of our industry. One is that much of that dollar goes to cover fixed charges on investment which does not vary much over good times and bad; hence, it is possible to maintain a more stable price for the service throughout the business cycle. For example, the only change in our rate since 1932 has been that resulting from a coal clause providing for recovery of the increase in the price of coal used in the plants. The other significant feature is that with a ratio of investment to revenue of 6 to 1 where a high load density area is supplied, it is not difficult to see that as lower load density areas are invaded, either cheaper distribution methods must be found or higher rates must be charged if a 6 to 1 ratio is to be maintained. To do either requires all the skill the men of our industry can muster.

What I have given you is a brief picture of some of the features of district heating from the industry point of view. Now what are some of the features from the customer's point of view.

District Heating from Customer's Viewpoint

If you were a prospect being approached by one of our sales representatives and you were told that there is available to you a complete heating service 24 hr a day, 365 days of the year, which is ready to use at the touch of a valve or the click of a thermostat, that it requires no bulky equipment using up valuable basement space, that you do not have to spend your valuable time looking after it, that you do not even need a stack in the building to install it, that the service is piped into your building at no expense to you, and that the service is metered and you are billed only for the amount of steam you use, your reaction would be only one of two things: either "What are we waiting for" or your suspicion is aroused to the point where you say, "What does it cost?" They are both good questions.

COMPARISON OF COST

Obviously we cannot manufacture ready-made heat and deliver it to your door for the same price that some coal dealer can dump a pile of coal on the sidewalk in front of your shop. So just as there is no comparison in the service rendered in either case, just so there is no similarity between the price of steam service and the price of the coal pile. I believe most of you realize though that coal is not the entire cost when one provides his own heat. A true comparison between the cost of private plant operation and that of district steam should take into consideration all of the other items of expense which are eliminated with the use of such a heating service. What then does a cost comparison look like?

Table 1 shows a typical cost comparison. It represents an average customer who pays, let us say, \$3000 per year. This average revenue figure represents annual steam sales of about 2630 M lb when distributed over the heating season in proportion to the normal amount of cold weather.

At 7½ lb of steam per pound of coal, it would require 175 tons of coal if he were making his own steam. At \$8½ per ton, this is exactly \$1488. In other words, this is only about half the steam bill, and even assuming that the prospect recognizes many of the advantages of district heat, it would still represent an appreciable premium if that were the actual difference in cost. But let us look at these other figures for a moment. In addition to coal, he must pay for ash removal. This is estimated at \$200. He has stoker and boiler repair bills every year. Well, perhaps not every year, but over the life of the equipment the repairs may be conservatively estimated at \$125 per year. There is a certain amount of makeup water that has to be added. That is \$15 per year. It costs him \$75 a year for electric power to operate the stoker and to provide the necessary light in his boiler room. Since it is a fair size building, it requires a sizable boiler plant which requires more than casual attendance. Assuming that a man is needed for this purpose only nine months out of the year, labor may be conservatively estimated at \$1200. There are also certain additional insurance costs which must be paid. They are estimated at \$36 per year. These total up to \$3139 a year. But that is not necessarily all. If he is not already at the place where he needs a new boiler, sooner or later he will be. The comparison then is not complete until he includes fixed charges. On an estimated investment of \$7500 the fixed charges at 12 pct are \$900 per year, and when this is added to the annual operating expense, we find the total annual charges against the plant to be \$4039. And when this is compared to a \$3000 annual steam bill, there is an annual saving of \$1039 in favor of district steam.

But suppose he says, "The boiler is in good shape. My lease has only five more years to run so fixed charges mean nothing to me." So he automatically drops out \$900. And suppose he says, "Firing is only an incidental job for my janitor who has a dozen other things to do at the same time. Even if I eliminate the boiler, I could not

eliminate the man so I cannot save $1200.

Thus if he drops out another $1200, in his mind at least his heating cost is $1939 or approximately $1000 less than district steam. Labor and fixed charges are two of the largest items that go towards tipping the balance in our favor. However, in many cases, although labor and fixed charges are real, they may not have a sufficiently direct bearing on the problem to secure recognition by the prospect. Under such conditions, we have a tougher selling job for it is then that we have to convince the prospect that the greater comfort, freedom from worry, a cleaner basement space for other uses, and perhaps several other items is worth far more than the extra cost of steam service over plant operation. In most cases there is usually one or more features about district steam that are particularly attractive to the prospect. Some time ago I went back over the records of the jobs sold during 1946 for the purpose of determining what factors made it possible to arouse interest in our proposition.

FACTORS INFLUENCING PROSPECTIVE CUSTOMERS

Table 2 shows the results of such analysis. We discovered that there were nine major factors which made it possible to get our foot in the door and contributed greatly to successful negotiations. In many instances several of these factors influenced each sale. But in every case we can say that one of these was the major consideration affecting our negotiations. Note that out of the 49 customers sold during the year, "change of ownership or occupancy" was the primary factor in the greatest number of instances. Usually when a property changes hands, the new owner is full of enthusiasm to rehabilitate, improve and dress up the building because he has a new interest and a great ambition to make his newly acquired property outstanding. District steam service fits in perfectly with such a plan, and we emphasize that point in our discussion. Next in importance comes the enforcement of the smoke ordinance. This analysis, by the way, covers the first year after the smoke ordinance really became effective, and obviously it played into our hands to a greater extent than normally that particular year. In fact, it still plays an important part but only where previously satisfactory plants are now failing to comply with the ordinance through deterioration of the plant or relaxing of operating supervision. Next comes the factor of "desire for improved conditions" which becomes more real as business conditions improve. Where people have more money to spend, they are willing to put more emphasis on comfort and convenience. Next is "building being remodeled." This is in line with the first factor that we discussed except that there is no change of occupancy or ownership involved. It is not necessary to discuss all of these factors. You can see the relative importance of the others. I shall comment briefly on "new buildings." Almost without exception, new buildings are rather easy to sell. For when a building is yet in the drawing board stage, it is much easier to eliminate boiler plant investment, to plan the use of basement space, and to save the investment in a stack. This factor is eighth only because there was little new building in the district at that time. Strange as it may seem, but much to the satisfaction of those of you in the coal business, anxiety over fuel supply did not appear to be critical. Apparently there was more confidence in the coal industry's ability to prevent serious trouble than was indicated at times by news items.

Table 1 . . . Comparison of Annual Cost Private Plant vs District Steam Service

(Average size customer—A.C.S.H.C.)
Estimated annual steam requirements, 2630 M Lb

Estimated private plant costs:		
Coal (175 tons @ $8.50 per ton)	$1,488.00	
Ash removal	200.00	
Repairs to boiler and stoker	125.00	
Water	15.00	
Electricity	75.00	
Labor	1,200.00	
Insurance	36.00	
Total annual operating cost	$3,139.00	
Fixed charges (12 pct of $7,500)	900.00	
Total annual charges		$4,039.00
Estimated annual cost of district steam		3,000.00
Annual saving favor district steam		$1,039.00

Table 2 . . . Relative Importance of Certain Considerations Which Made Possible the Opening of Negotiations Resulting in Sales during 1946, Allegheny County Steam Heating Co.

	Number of Customers Secured
1. Change of ownership or occupance	11
2. Enforcement of smoke ordinance	9
3. Desire for improved conditions	7
4. Building being remodeled	6
5. Original supplier eliminated	5
6. Necessity of boiler replacement or plant difficulty	4
7. Saving in annual operating cost	4
8. New buildings	2
9. Anxiety over fuel supply	1
Total	49

Summary

That briefly is a general picture of some of the problems and features of district heating here in Pittsburgh. I should say that so far as the industry at large is concerned, what applies here applies nearly the same throughout the entire industry. Some of the features may differ in minor detail but generally they are the same. If you were to talk to a great many of the district heating men, I believe they would tell you that the largest single problem facing the industry today is how to expand their systems at a cost which can be justified by the business to be secured, and when I say business secured, I mean not only the magnitude of steam sales but also business secured at a price which from the prospect's point of view is commensurate with the value of the service.

In regard to the size of our industry, I should say there are at present several thousand district heating systems including those for institutions, individual housing projects, and groups of public buildings. Of these, about 275 represent operations of a commercial nature run by utility companies. Of these 275, roughly 50 of the largest do most of the business and from a company membership standpoint, comprise practically the entire support of the National District Heating Association which is an organization for the advancement and promotion of the district heating business. These 50 large companies have a total investment of approximately 186 million dollars. Their sales amount to about 50 billion pounds of steam annually for which they receive roughly 53 million dollars annual revenue from 27,000 customers. It will be of interest to you that these steam sales create a market for some 2½ to 3 million tons of coal a year.

Nearly every large city and many smaller ones in the United States, except in the Southern part of the country, has a district heating system of which New York City is by far the largest. For example, New York sells approximately 15 billion pounds of steam a year to 2300 customers. This is ten times the sales of our Company here in Pittsburgh which sells a billion and a half pounds of steam annually. Next largest is the one in Indianapolis. The third is Detroit; the fourth, Cleveland; and the fifth, Rochester.

Frequently we have persons say to us, "That certainly is an excellent serv-

ice. When are you going to have it available for my home?" Maybe some day we will. In some cities there now is some residential district heating such as Virginia City, Minn., where the entire city uses district steam. There are two systems in use for residential heating near Philadelphia. The advisability of supplying a given territory depends upon a careful analysis of many considerations. Not long ago Battelle Memorial Institute undertook a study for Bituminous Coal Research, Inc., to determine the advisability of using district steam on a broad scale for residential heating. I am not sure that their report is yet completed, but I am informed that their study shows that such projects are advisable only under the most favorable circumstances.

Future Outlook

Now just where does the industry stand today and what does the future appear to hold for it? The business districts of most of the large cities are now fairly well developed although there is considerable room for expansion in the fringe areas of these districts and in some industrial areas. There is also the possibility of developing a greater summer load through the use of steam for summer air conditioning.

Several things have happened in the last few years to improve the status of our industry and perhaps make it possible to expand more rapidly. First, the price which we must charge for the service now compares more favorably with the actual out-of-pocket expense of operating a private plant. Contributing to this condition is the fact that the basement space is now more valuable, investment costs in plant equipment are higher, and operating costs all along the line have increased considerably. All of these tend to boost private plant costs to an all time high. On the other hand, steam service charges have not and very likely will not increase in any such proportion. The reason for this is that a large portion of the selling price is to cover fixed charges on investment which remain practically constant.

The second thing favorable to the industry is a greater appreciation on the customer's part of the benefits of district heating. When business is good and profits more favorable, people spend more for the comforts of life. Since district heat offers many advantages in this direction, it secures a greater acceptance now.

The third factor is a greater consciousness on the part of the people of the community of the civic value of district heat. We have become increasingly conscious of the economic loss due to smoke. Large district heating plants are operated efficiently and smokelessly. The tendency is for small individual plants to be operated inefficiently, and many of them produce large quantities of smoke and dirt. With traffic increasing almost daily, the importance of avoiding traffic congestion takes on a new significance and the elimination of many trucks handling coal to dozens of buildings in congested areas is important.

The last factor that has served to improve the status of the industry is a general improvement in operation. Time always develops new gadgets, new methods, new devices to improve existing systems or ideas. There have been improvements in efficiencies, both of a general nature and particularly in the matter of distribution. I should say also that there has been greater organization of selling efforts to develop new load. We have become more customer minded, so much so that many of the steam companies maintain a department for the sole purpose of helping the customer solve his heating problems and to operate his system in the most economical manner.

There are now four rather distinct fields which offer possibilities for load building. These are: (1) congested business and residential districts, (2) districts of individual family residences and housing projects, (3) grouped industries, and (4) institutions. Congested business districts represent the most favorable field because of the high load density which I have already mentioned. While there are a number of district heating systems supplying detached residences, in many instances the high cost of the distribution system and operating costs compared to the small quantities of heat required make the cost of district heating service high compared with other methods of heat. On the other hand, certain housing projects lend themselves much more favorably to district heating systems because of the higher load density and also because the control of ownership is under one hand. This is not usually true of individual family residences.

Some of you may have read of Peter Cooper Village and Stuyvesant Town in New York City, both projects of the Metropolitan Life Insurance Company's postwar development. Thirty city blocks of tenements and antiquated industrial and commercial structures are being rehabilitated into first class housing facilities for 12,500 families. The New York Steam Corp. will supply their entire steam requirements.

Supplying steam to industries for manufacturing processes and for space heating shows promise for future development. Many industries require large quantities of steam, and where they are advantageously grouped, it is often desirable to purchase steam as well as electric service from the local utility. Institutions such as Carnegie Institute of Technology, Pennsylvania State College, University of Pennsylvania, and others, are examples of district heating in this field. In Pittsburgh we have a rather large district heating operation of an institutional nature in the Bellefield Boiler Plant, which supplies a group of buildings of a noncommercial nature in the Oakland district. They include Carnegie Institute, Cathedral of Learning, Mellon Institute, Presbyterian, and Eye and Ear Hospitals, and perhaps some others. I understand it is a nonprofit venture where each user shares the total expense in proportion to the amount of steam used.

District heating is not peculiar to the United States since there are systems also in Alaska and Canada and in several countries in Europe. England, France, Belgium, Germany, Russia, Czechoslovakia, all have district heating systems.

Now I must admit I have tried to cover a pretty big subject in a limited amount of space. Obviously I could give you only a superficial picture of our small but proud industry. I hope, however, that this will give you a better appreciation of what our industry is and does, what some of its problems are, and most of all how valuable the service is which we render to the public.

Sinking with the Hydro-mucker at Mather "B" Shaft

By J. S. WESTWATER,* Junior Member, AIME

The Mather mine of The Negaunee Mine Co. embraces nearly all of Sections 1 and 2, T47N, R27W, within the limits of the cities of Negaunee and Ishpeming on the Marquette iron range of Michigan's Upper Peninsula. Operations of the Negaunee Mine Co., the stock of which is jointly owned by Bethlehem Steel Co. and The Cleveland-Cliffs Iron Co., are under the direction of the latter company.

The opening of the Mather mine was described by C. W. Allen.†

Mather "A" shaft was completed to an initial depth of 2352 ft in early 1943. The shaft capacity of over one million tons yearly is in prospect of being realized by orderly development and mining, and sinking of the second or "B" shaft then started in accordance with previous planning. Mather "B," also a vertical shaft, was designed as a twin to Mather "A" shaft and the sinking technique developed at Mather "A" was utilized in "B" shaft operation.

Shaft Sinking General Plan

An average depth of 16 ft of sand overburden at the shaft site was cleared away by bulldozer, and the shaft approximately 15 by 20 ft in rock opening put down to an initial depth of 87 ft. Mucking of the broken rock to this depth was accomplished by an Insley excavator equipped with a ½ yard capacity Owens clamshell bucket. Addition of 200 lb weight to each lip of the clam bucket provided adequate "bite" to dig the broken diorite and jasper rock. Temporary heavy wood bearers were placed across the shaft mouth to support the steel shaft sets which were hung in this initial 87 ft depth, aligned accurately and concreted in place from the lowest set upwards. The permanent steel headframe foundations and three service tunnel openings to the shaft were also concreted in the upper section to the shaft collar.

A temporary wood headframe was then erected over the shaft mouth, and initial hoisting machinery assembled consisting of a Lake Shore 12 ft diam single drum grooved for a 1⅜ in. rope and powered by two 400 hp motors to attain hoisting speed of 1100 fpm. Compressed air was obtained from a Nordberg 2 stage tandem horizontal type with a capacity of 1500 cfm. An enclosed landing platform at the shaft collar was equipped with a rotary dump and pocket for discharge of the rock into a Euclid 15 ton truck.

On completing assembly of this equipment and installation of a special cage in the shaft, sinking was resumed on a three shift, six days per week

San Francisco Meeting, February 1949.

TP 2596 A. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before July 30, 1949. Manuscript received Jan. 26, 1949.

* Superintendent, Mather Mine "B" Shaft, Cleveland-Cliffs Iron Co., Negaunee, Mich.

† C. W. Allen and L. C. Moore: Opening the Mather Mine. *Trans.* AIME (1945) 163, 293.

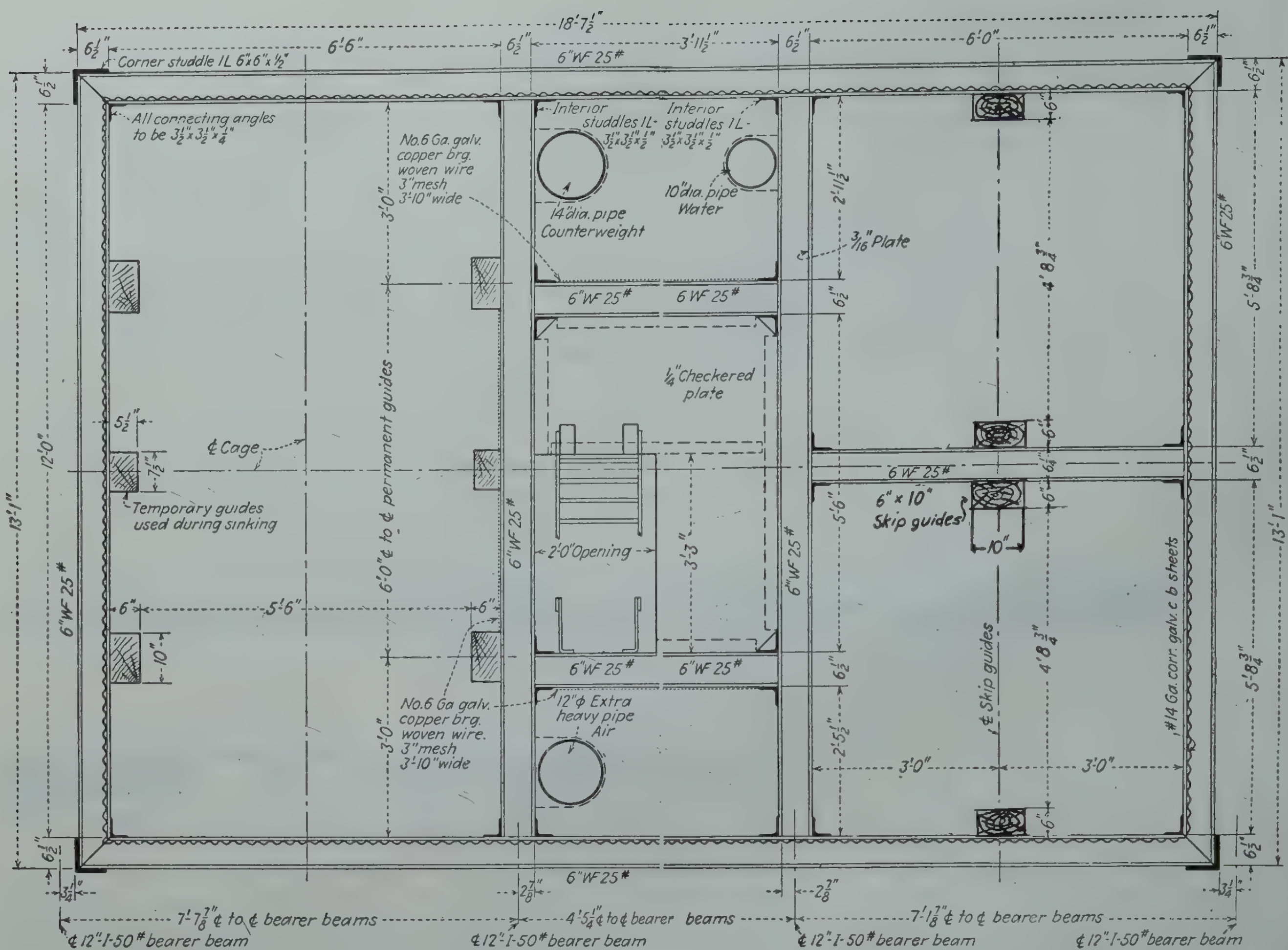

FIG 1—Steel shaft sets.

schedule. The equipment used and sinking procedure established in the Mather "A" project was fully covered by C. W. Allen in his earlier paper. Mather "B" shaft, while re-using the same or employing similar equipment and following generally the pattern adopted for "A" shaft, has made use of the advantages offered by the tungsten carbide tipped drill bit and of a hydraulically operated clamshell bucket in mucking operations.

Fig 1 shows the arrangement of steel shaft sets which are installed on 7 ft centers. The sets are hung as shown from 6 by 6 by ½ in. angle studdles and are blocked by a reinforced concrete ring, bevelled below the set to the rock walls, or by sacked concrete jammed between the beams and the rock. Normally every other set is placed with a bevel concrete pour and the alternate sets blocked with the concrete filled sacks, however in sections of the shaft possessing unstable wall conditions, each set is installed with the bevel pour.

Fig 2 illustrates sets that are of Bethlehem Mayari-R steel, 6 in. 25 lb *H*-beams, and are credited with a corrosion resistance 6 times that of ordinary steel. The arrangement provides for a 4 compartment shaft consisting of a cage compartment 6 ft 6 in. by 12 ft 0 in. inside, a ladder and pipe compartment 3 ft 11½ in. inside and 12 ft long, and 2 skip compartments 6 ft 0 in. by 5 ft 8¾ in. The cage and skip compartments are separated by the ladder compartment. In addition the skip roads are completely sealed from the other two compartments by a partition consisting of $^3\!/_{16}$ in. steel plate which extends the width of the shaft and is made air tight to the shaft walls by the compaction of concrete filled sacks between the edge of the plate and the rock. In normal ventilation the cage road is downcast with fresh air and the skip compartments upcast carrying the discharge air to prevent freezing of the skip dumps. In case of fire underground no contamination of the fresh air entry is possible with the plate seal. Corrugated lath, No. 14 gauge, copper-bearing galvanized sheets, fully enclose the shaft perimeter. The sheets are made fast to the 6 in. *H*-beams set by bolting with a specially designed clip.

In the ladder compartment, wire mesh of No. 6 gauge, 3 in. square openings, is fastened to the double cage guides, extending wholly across the opening of the ladder road proper. At the sollar sets of the ladder road, 2½ in. guard angles are placed across the interior studdles to prevent mis-step.

Two pairs of wood guides 6 by 10 in. will be installed in the cage compartment to serve the permanent aluminum double decked cage, replacing the 5½ by 7½ in. single guides now serving the sinking cage. The same dimension wood runners will serve the skips which will be equipped with rubber tired guide wheels to minimize wear of the supplementary steel guide shoes.

The ladder compartment is divided into three parts, the center serving as the ladder road proper, and each end

compartment for pipe and power cable installation. A 14 in. cage counterweight pipe is being installed as sinking progresses and serves as ventilation conduit from a size 45 Sturtevant planovane fan which is capable of producing 3000 cu ft of air per minute through 2000 ft of 14 in. pipe against a maximum static pressure of 20 in. of water. An adjustable by-pass arrangement is provided between the fan and the shaft conduit to permit blowing or exhausting action of the fan which rotates in one direction only. A 10 in. diam water discharge line and a 12 in. diam compressed air line are to be installed, and the weights of these large diameter pipes are carried only by the bearer sets installed at intervals of 150 ft. These bearers are 12 in. 50 lb *I*-beams and 4 beams constitute a set. They are placed across the short dimension of the shaft, two immediately adjacent to the end pieces of the shaft sets and two over the interior dividers. These bearers are of sufficient length to jut into niches cut in the shaft walls. The *I*-beams are bolted to the shaft set and the combination is then embedded in a solid ring of concrete.

A sinking cage, shown in Fig 3, has a platform 5 ft 6 in. by 9 ft 0 in. and is equipped with safety catches and a steel crosshead which permits lowering to a distance of 30 ft below the bottom shaft set. The use of this type cage permits movement of men and supplies with comparative safety and in the mucking operation, hoists a 2¼ cubic yard capacity car. In conjunction with the cage is a 2 yard capacity mucking tray (Fig 4), equipped with a bail and possessing two hooks at the front, which is hoisted by an Ingersoll-Rand K4U air hoist having a rated capacity of 3500 lb rope pull at 80 lb pressure and a lift speed of 95 fpm. The cage is lowered to the muck pile and the hooks of the tray engage two dumping bars on the cage permitting the tray to discharge its load through an opening in the side of the cage into the car blocked in place on the cage platform.

Details of Sinking Operation

DRILLING

Iron formation or jasper was penetrated from near the collar to the 1650 ft depth. A 7 ft depth of cut was taken consisting of 72 holes drilled breaking to a *V*-cut. The drill machines used are the Ingersoll-Rand J-50 Jack-

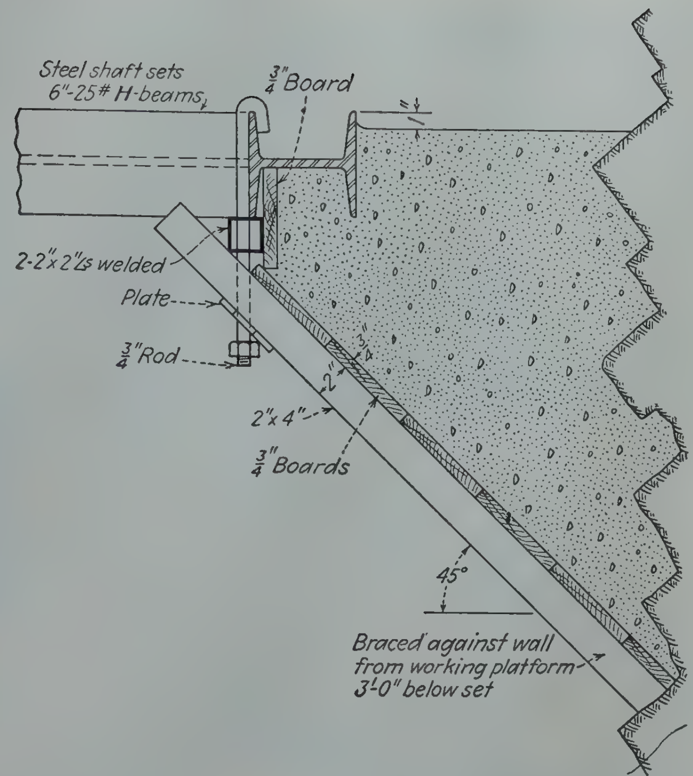

FIG 2—Method of concreting steel shaft sets.

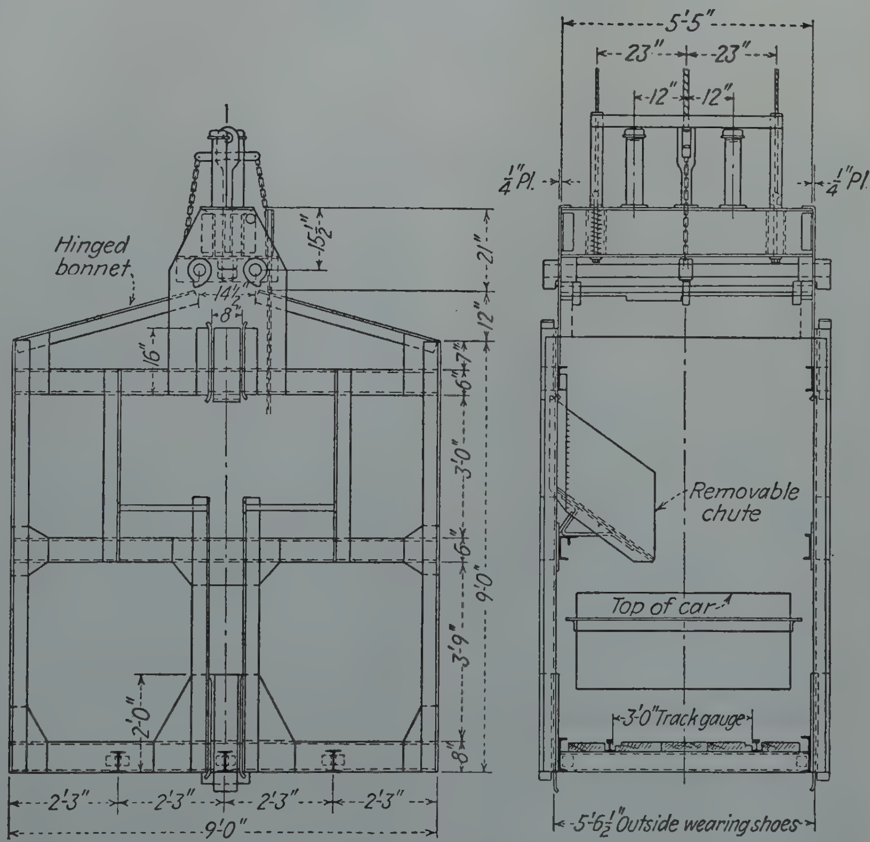

FIG 3—Sinking cage.

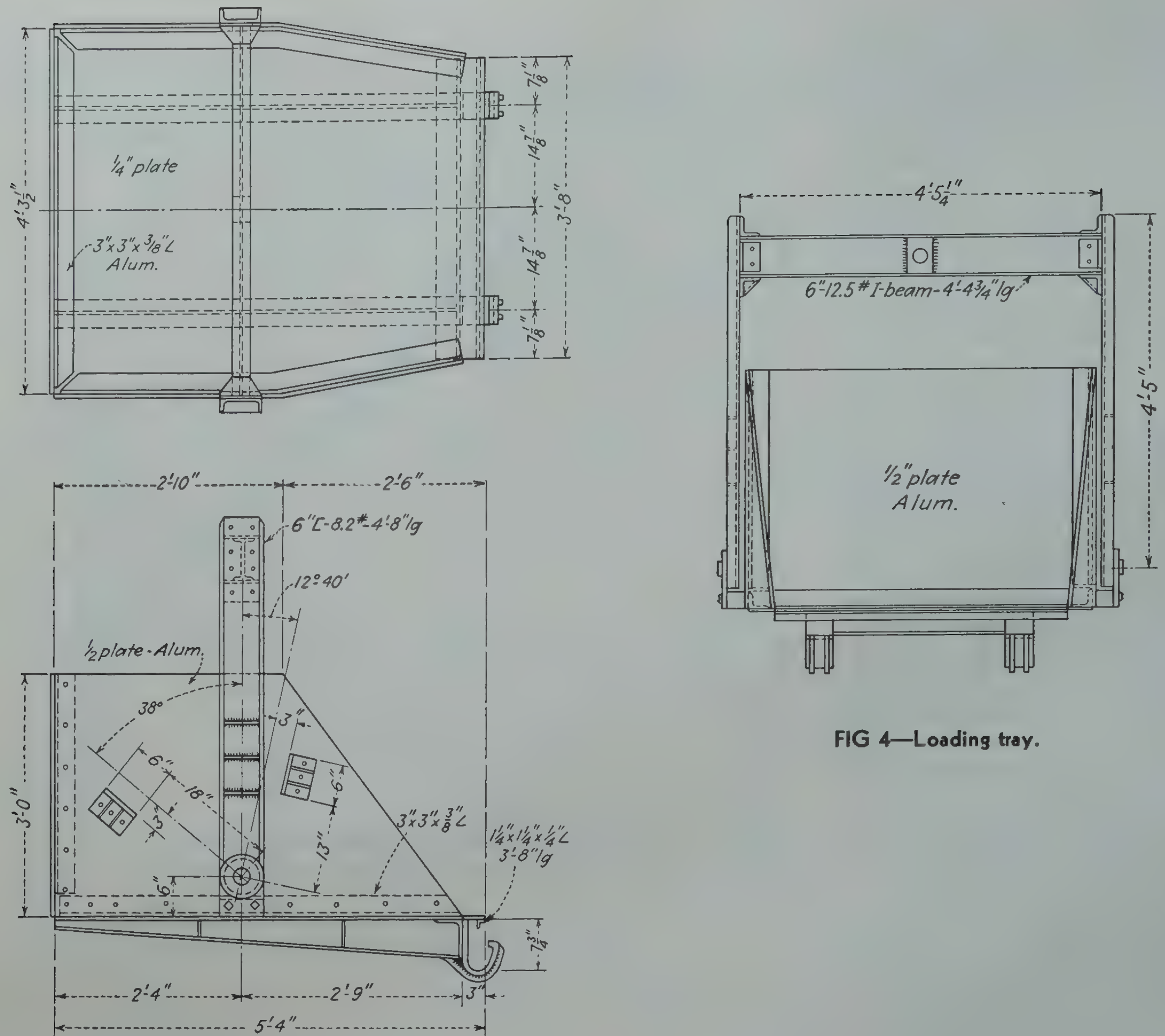

FIG 4—Loading tray.

hammers of the 60 lb class equipped with automatic air-water backheads.

One inch, quarter octagon, Bethlehem high carbon drill steel is used. The bits employed during the first several months were the Ingersoll-Rand jackbit 2¼ in. gauge starter. Drilling time using 10 machines and making a 72 hole round, averaged 4¼ hr. Tests made with the hard metal insert bit, Series 13 CS4 Carset Jackbit 1½ in. gauge, were immediately successful and the entire drilling procedure was then placed on full employment of this type bit. Average drilling time with the hard metal bit was 2 hr with an occasional 72 hole round being finished in actual drilling time of 50 min. In Mather "A" sinking, an occasional layer of hard blue jasper would boost drilling time tremendously, one example being a 4 ft cut that required the use of 1700 jackbits and 11 hr to drill. Mather "B" shaft is in the same geological horizon and has penetrated similar hard layers of blue jasper and has, since the adoption of the tungsten carbide bit, completed drill rounds in a maximum time of 3½ hr. At the drilling of the maximum time round, a depth of hole of 8 ft was made. Bit life of the standard jackbit averaged 2 ft per sharpening whereas the Carset bit produced 135 ft before discard, the predominant cause of which was gauge loss.

Early drilling with the Carset bits, employed alloy steel rods but trouble was encountered with connection studs working loose, and inasmuch as the previously used high carbon steel had performed satisfactorily the studs were driven in high carbon drill rod entirely and the rods and bits have given excellent service. It is the practice to place bits on the rod change lengths in the mine shops and securely tighten them in place using a 24 in. Stillson wrench. Sufficient extra rods are on hand in case of breakage during drilling, eliminating any necessity for bit changing by the miners during the round. The drill machines, rods, and bits are thoroughly inspected after every drilling period. In the case of bits, a gauge test is made and if found to have lost to the extent of 1/16 in. the bit is removed. In the case of longer lengths, a bit is interchanged from a short length and a new bit placed on the short lengths. This ensures greater gauge on the shorter rods and has eliminated to a great extent loss of time due to stuck steel.

Bit discard is due preponderantly to gauge loss which is quite severe in down holes drilled in the extremely abrasive, high silica jasper formation. It has been found, however, that the bits removed from service because of gauge loss produce a footage equal to that given in the shaft when used later in drilling horizontal or upper holes at other company mines.

Air was supplied to the machines in early drilling by means of a header equipped with 12 I-R line oilers of 1 pint capacity, and necessary air and

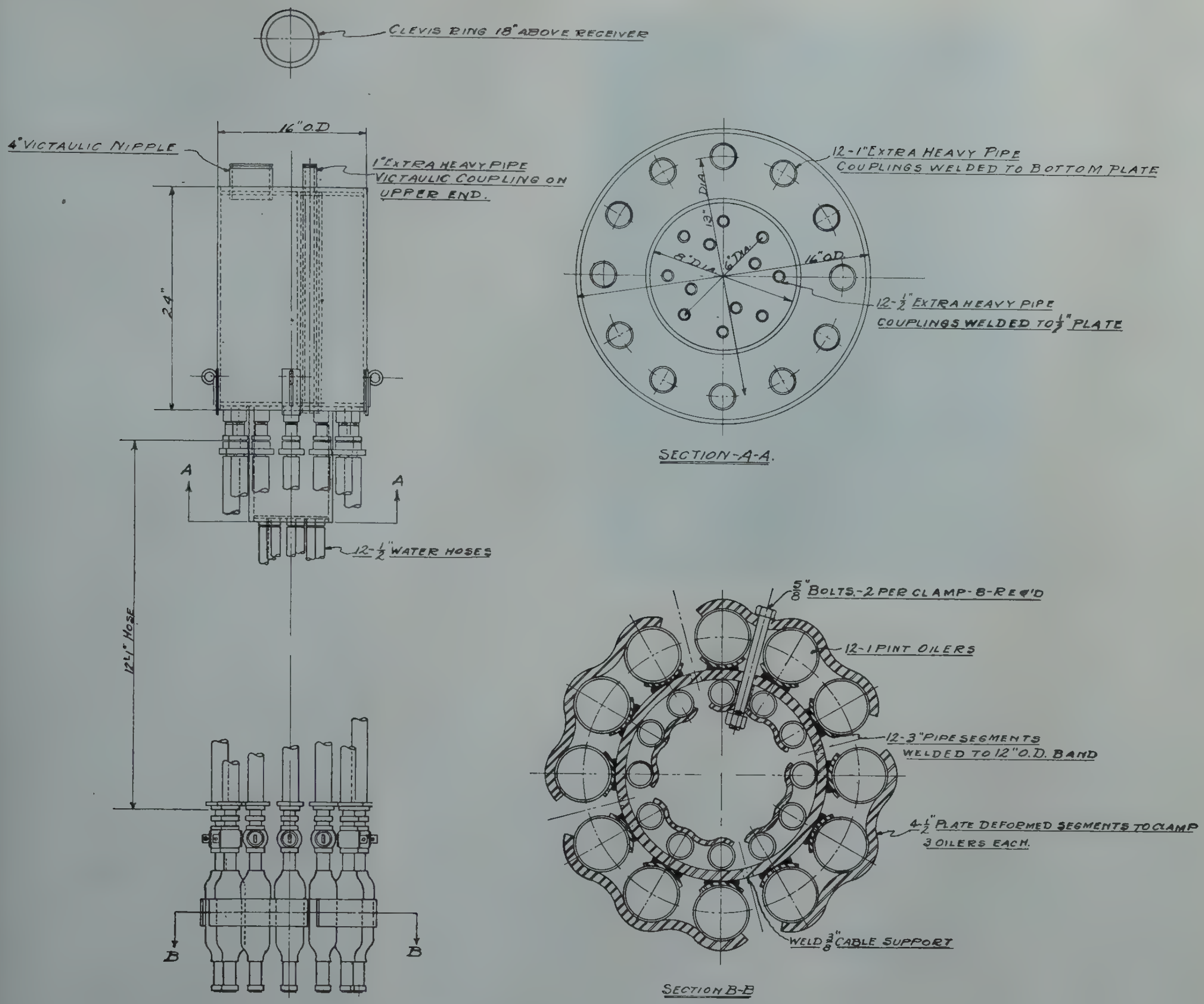

FIG 5—Drilling header for shaft sinking.

water connections. The header was fed by a 6 in. line and a connecting 3 in. rubber hose. A K4U utility hoist mounted on a shaft set raised and lowered the header as desired. From the header, air was supplied to the drill machines by 30 ft lengths of 1 in. hose to ensure adequate volume and pressure. It was found, however, that insufficient lubrication was being given the drill machines, and that the 1 in. hose was awkward and cumbersome in handling the machines, therefore, a header or manifold of different design was built at the mine shop. This header as shown in Fig 5 consists of an air receiver, 16 in. diam pipe 24 in. long, through which passes, completely enclosed, a 1 in. extra heavy pipe for water. Twelve 1 in. air hoses are connected to the bottom of the air receiver near the outer circumference. Likewise twelve ½ in. water hoses lead from the water receiver a section of 8 in. pipe, 12 in. long, welded to the bottom side of the air receiver. These air hoses are 12 ft long and connect to 12 oilers of 1 pint capacity securely fastened in groups of three, in a circular arrangement 19 in. od. As in the upper connection the air lines occupy the outer ring and the water lines the inner circle. Air hoses of ¾ in. lead from the oilers to the drill machines. The lengths of these machine hoses vary from 12 to 17 ft so that when the header is suspended over the shaft bottom just above head height, the corner machines have sufficient length. This octopus type header, designed by mine engineer Harry Swanson, has delivered adequate air, water, and lubrication and has as its strong feature a shape which provides no difficulty in traversing the constricted pipe compartment of the shaft when being raised out of the way of the blast.

The number of holes drilled per blast has varied from 65 to 85 matching the ease or difficulty of obtaining satisfactory fragmentation and cleanly broken bottom. It was found in the sinking of "A" shaft, which had its long dimension parallel to the strike of the formation, that the slates dipping at 45° would spall and slough to a semicircular arch especially on the upper side of the dip. Not infrequently that side would be drilled at the corners and the remaining holes pulled away from the proposed rib outline almost to the middle of the shaft. Despite this buffer approach the severe arching would repeat. To counteract this weakness in the slates, Mather "B" was turned so that the short dimension of the rectangular outline was presented to the strike and greatly reduced overbreak has been experienced.

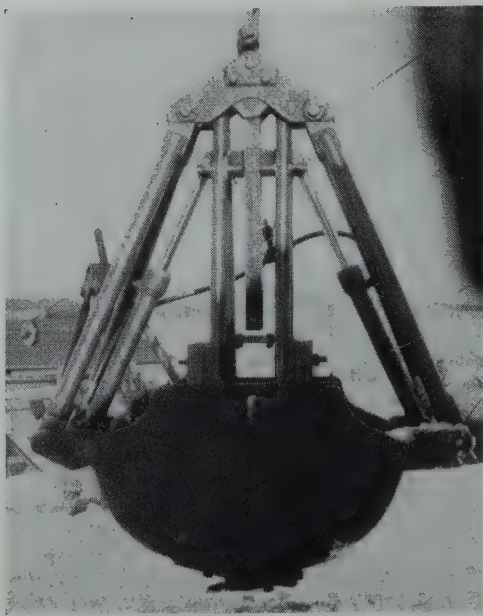

FIG 6—Hydro-mucker.

BLASTING

On completion of the drill round, the holes are blown clean with compressed air and charged with gelatin dynamite in the ratio one third 90 pct and two thirds 80 pct Hercules gelatin extra in cartridges 1¼ by 8 in. It had been determined by trial that the 90 pct dynamite in the bottom of the holes produced a cleaner blast bottom in the dipping layered jasper formation. The primer cartridge is loaded as the second stick in the hole, and contains a delay electric blasting cap. These delays range from the three available short period or millisecond delay to the standard 12th delay. Lead wires from the individual blasting caps are bunched in groups of not over 5 by color and are twisted in a tight connection to one of two bus wires suspended by wooden pegs directly over the blast area. These bus wires are tinned, bare copper No. 14 gauge wire. A No. 14 duplex wire leads above the blast area to connect to a No. 4 rubber covered Tyrex cable which extends the length of the shaft to a blasting station in a concreted service tunnel near the shaft collar. The blast is fired from a locked safety switch, the key of which is in the possession of the shaft boss. Alternating current is used to fire the round, 110 v was used for the first 1600 ft and 220 for the succeeding lift. The junction of the No. 14 Duplex to the No. 4 is made on a box equipped with wing nuts for easy manipulation. Before the junction is made the shaft boss, coming up from a last inspection of the blast wiring, tests the No. 4 Tyrex for stray current by placing the lead wires of a squib across the Tyrex ends. The connection is then made and the boss continues to the shaft collar to fire the round from the locked switch which is shunted in the open position.

It was found in the initial drilling with the hard metal bits that because of the uniform small diameters of the holes, 1½ in. as compared to the standard jackbit inverted tapered hole ranging from 2¼ in. at the collar to approximately 1¾ in. at the bottom, much less dynamite could be loaded in the 1½ in. hole. The ratio was approximately 50 pct. On blasting however, equal if not better fragmentation was obtained from the smaller powder loads. One explanation offered for this seemingly unbelievable result is that perhaps because of the uniformly parallel sided hole of the hard metal bit, the gas pressure is retained slightly longer than in the outward sloping conical hole produced by the larger jackbits.

In the hand mucking stage of sinking, good fragmentation was obtained in the jasper with a 7 ft cut. Use of the hydro-mucker eliminated the necessity for fine fragmentation and the cuts were deepened to 9 ft. Wall stability is now the governing factor in depth of round blasted.

MUCKING

Company established sinking practice insisted for reasons of safety that in deep shafts a cage be used to hoist shaft rock rather than the guided sinking bucket or skip and trap door arrangement commonly used. Although there has been no shortage of volunteers for shaft sinking with its attendant laborious hand mucking, we have long searched for some type of mechanical mucking that could profitably be used in lightening the load for the crews and at the same time speed the sinking operation. Shaft mucking was about the last phase of underground operation within the company that had not been mechanized in whole or major part.

The combination of tray and sinking cage, which at times filled almost 40 pct of the area below the last steel set together with unset concrete supporting the last set, precluded the employment of any heavy traveling form of digging equipment. It was believed, however, that a type of clamshell bucket suspended from a fixed point up in the shaft where the concrete had reached a good setting strength, could be swung pendulum fashion to all points on the muck pile in this 15 by 20 ft area at the bottom. Also it was desired to use a mechanical method that would be comparatively simple, free from appurtenances, and fit in with the equipment of the hand mucking tray system. If, as mechanical innovations are prone to do, failure of some part should occur, especially in the early stages, the shaft mucking could proceed in the previously accustomed fashion with little interruption.

In late 1947 trials were made with several types of industrial rope-reeved clamshell buckets actuated by various types of air and electric powered hoists. These experiments were made in a full scale model of 4 shaft sets on surface to simulate actual space conditions in the shaft and to work out any difficulties that could be foreseen before placement of the apparatus in actual service. Mechanically, the result of these trials was successful, but it was obvious that some training of the miners as clamshell operators would be necessary for successful combination of the placing and digging action of the equipment. At that time, however, acquaintance was made with the Milwaukee Hydrocrane Co., who utilized a bucket operated hydraulically in conjunction with their mobile crane. This

type of bucket, it was apparent, would eliminate the drawback of a rope-reeved clamshell pulling away from the muck pile, and, further, the bucket would be comparatively light in weight for easy movement by the miners and yet obtain a load without the restrictive directional pull of the digging line.

With the Hydrocrane people offering fullest cooperation, a layout was made which was very simple in design, adding only the bucket and a hydraulic power unit to the equipment already in the shaft for hand mucking purposes. The bucket as manufactured by the Milwaukee Hydrocrane Co., now the Hydrocrane Division of Bucyrus-Erie Co., was of ⅜ yard capacity and operated by a single hydraulic ram of $2\frac{9}{16}$ in. diam and 19 in. stroke mounted vertically within the bucket yoke or frame. The bucket in dimension and outline is very similar to the conventional industrial rope-reeved bucket and weighs 1350 lb complete (Fig 6). A low head type of bucket equipped with two horizontal rams was being developed by the Hydrocrane people which seemed to offer greater advantage for shaft work with its lower center of gravity, but to initiate the hydro-mucker and work out the weak spots bound to appear, it was decided to use the vertical type rather than wait for satisfactory development of the low head bucket.

Hydraulic power for the bucket was provided by assembly at our General Shops of a unit consisting of a 20 gal oil reservoir tank, a small filter, a 15 hp motor, two hydraulic pumps and a control bank, all mounted within a base and bail to permit lowering through a pipe compartment 2 ft 5½ in. by 3 ft 11½ in. Fig 7 shows the original power unit built. A 15 hp, 220 to 440 v, 3 phase, 1800 rpm, drip proof motor mounted on a base plate drives two Blackhawk P-197 pumps by *V*-belt and pulleys. These 3 gpm pumps, mounted on brackets for quick belt adjustment or removal, are 6 cylinder radial style piston type pumps, wherein radial motion is transferred into reciprocating motion by means of a wobble plate. The pump is capable of pressures up to 6000 psi. Good bucket digging action and pump life have been obtained from operation of the pumps at 5000 lb pressure and a speed of 1400 rpm. Bearings of the pump are antifriction type, throughout. At present an improved type of pump classified as P-237 is being used. This latest design has eliminated several causes of failure in the earlier model and contains an improved piston spring, removable valve inserts, Amco metal rocker caps and swivel sockets on ends of the pistons and uses no soft packings. At the outset, the rigorous demands of shaft mucking which comprised practically a straight 17 hr usage out of a total 27 hr cycle, the P-197 pumps failed in four to five days service. This was to be compared to an average year and a half life for the same type pump used in the mobile crane. This feature was extremely critical and at first jeopardized an otherwise highly successful operation. Thorough and constant observation by mine mechanic, John Nigra, singled out the prime causes of failure and with splendid cooperation from the Blackhawk Manufacturing Co., these weaknesses were rectified and since the two pioneer models of the P-237 were introduced into the shaft for proving ground purposes, no further trouble has been encountered.

FIG 7—Power unit.

The procedure followed in removing shaft rock with the hydro-mucker is essentially the same as that of hand mucking, that is, the 2 yard tray is filled by the clamshell bucket and the contents of the tray dumped through the side of the cage into the 2¼ yard capacity car. A major advantage of the hydro-mucker which should be emphasized is the extremely short preparation time necessary for the start of the mucking cycle. Since the tray and bucket are the only two pieces of equipment removed during other phases of the sinking operation, it is a matter of minutes to connect the hydraulic line and begin the mucking. A second advantage is the fact that at no time have maintenance or repairs consumed more than a few minutes during the mucking cycle. During mucking, the power unit is usually mounted on the 4th or 5th shaft set from the bottom. A 65 ft long, ⅜ in. id, high pressure hydraulic hose line leads from the control bank of the unit, over a stationary reel from which loops are paid off as additional length is necessary, down to the bucket. A miner sitting on the lowest shaft set, which is customarily 20 to 30 ft above the muck pile, opens and closes the bucket by manipulating two draw cords attached to a short crossbar or yoke on the control bank lever in the power unit above. He has a commanding view of the operation and is up out of the way. The bucket is raised or lowered by a reversible air hoist of the K4U type which is operated by a miner standing on the muck pile holding two draw cords attached to a yoke on the hoist control valve lever. A ¾ in. wire rope which raises the bucket, is double reeved by being passed through a 10 in. traveling block above the bucket, and is then dead-ended on the shaft set above that on which the air hoist is mounted.

When the tray is full, the bucket is swung to one side and permitted to dig itself into the broken rock to maintain an upright steady position. A removable pin is quickly slipped out of the clevis on the bucket yoke, and the rope

Table 1 . . . Monthly Progress Record

Month	Days Worked	Footage		Cuts Blasted	Steel Sets	Sets Concreted	Cubic Yards Concreted	Cubic Yards Rock Hoisted	Cubic Yards per Foot	Advance per Day
		Advance	Depth							
Prior 1947		128	128	19	15	12	494	640		
November 1947	26	88	216	16	13	7	33	1,446	16.4	3.4
December 1947	26	107	323	20	14	7	39	1,466	13.7	4.1
January 1948	26	142	465	26	19	9	41	2,210	15.5	5.5
February 1948	25	101	566	21	16	10	66	1,606	16.1	4.0
March 1948	27	118	683	23	17	17	125	1,900	16.1	4.4
April 1948	26	144	827	27	20	12	122	2,409	16.7	5.5
May 1948	25	138	965	26	20	13	44	2,480	18.0	5.5
June 1948	26	138	1,103	28	18	15	87	2,311	16.7	5.3
July 1948	26	140	1,243	28	21	12	82	2,680	19.1	5.4
August 1948	20	112	1,355	17	16	9	68	2,238	20	5.6
September 1948	24	153	1,508	19	22	11	105	2,810	18.4	6.3
October 1948	26	170	1,678	24	24	13	101	3,198	18.8	6.5
November 1948	21	138	1,816	17	19	10	95	3,054	22.1	6.6
December 1948	25	145	1,961	19	21	13	105	3,354	20.3	5.8

attached to the bail of the tray by the same pin and a similar clevis. The cage is lowered to the dirt pile and the tray raised. The position of the dead end of the tray hoisting rope, on the divider separating the cage and ladder road compartments, swings the tray to the cage allowing hooks on the tray to slide up along the side of the cage ensuring engagement on the dumping bars. After dumping its load, the tray is swung to a favorable position on the pile, the pin and block removed, attached to the bucket and the loading operation repeated. Three to four men swing the bucket into digging position and in the 15 by 20 ft area very little picking down is necessary since the straight sides and lip of the bucket fit the square corners of the shaft.

A 9 ft cut in the slates produces an average of 92 trays of 2 yards each or 185 yards of broken rock. All but ½ to 1 tray at the final clean-up is handled by the hydraulic bucket. It is advantageous to clean the bottom thoroughly before starting to drill, thus eliminating to a great extent hole-starting and stuck-steel difficulties caused by loose rubbly bottom. It was customary to pick and moil 2 to 3 cars to clean the bottom in the hand mucking stage, but the positive action of the 4 ton closing pressure of the clamshell teeth acts as a 5 pointed moil and seldom is it necessary to hand shovel more than ½ tray. Large slabs or blocks pried from the walls in trimming following a blast are easily handled by the bucket.

The original vertical bucket required 2 to 4 sec to close on a load and 13 sec to open. Trials with the double ram, low head, modified, materials-handling bucket duplicated the closing time and had a 6 sec opening time but this type bucket did not possess as good digging qualities in the rock as the upright bucket. Therefore, at the mine shops the two rams each 1¾ in. diam, 13 in. stroke were mounted inclined at 45° on the vertical bucket replacing the larger 2$\frac{9}{16}$ in. diam, single vertical ram to produce a hybrid bucket possessing good digging action and quick opening.

Performance data shows hand mucking production with 10 men as 8 yards per hour up to the time the hydro-mucker was introduced, which was August 1948. Since that time the yardage per hour has risen steadily as improvements were made and the system became trouble free and the month of December 1948 showed an output of 11 yards per hour which is an improvement in excess of 35 pct over hand mucking. A 10 to 11 man crew was necessary in hand mucking and while a 7 man crew is adequate with the clamshell, a 9 man crew is regarded as most efficient for all phases including drilling, shaft steel placement, and concreting work.

A monthly progress record, Table 1, gives information from which the general story of the sinking operation to date may be drawn. For example, January of 1948 had very good wall conditions requiring solid concreting of every other set according to plan and an advance of 142 ft was made. March had the same type of jasper rock from a hardness standpoint in drilling, but the blocky characteristic was much more pronounced, which factor demanded concreting of practically every set and required placement of 125 yards of concrete as compared to 41 yards in January. The cubic yards of rock hoisted per foot of advance was consequently high in March and the resultant gain was 118 ft as contrasted to the straight going 142 ft of January.

The comparison summary of shaft sinking figures between Mather "A" and Mather "B" shafts, which were put down through similar rock conditions and in which "B" used the same methods and equipment at the beginning, shows a marked increase in footage advance per month. The first year of sinking Mather "A" produced an average monthly advance of 99 ft, and during the second year an average of 107 ft was made with a monthly high of 124 ft of completed steel and concrete shaft. In the first year of Mather "B" sinking an average of 137 ft per month has been made with a high month of 170 ft on the three shift, six day per week schedule. It is further interesting to note that the average monthly advance of Mather "B" prior to the adoption of the hydro-mucker was 129 ft and in the past four months of clamshell usage the average was 151 ft of shaft advance. Since the sinking crews were of comparable caliber and of slightly less number in "B" shaft sinking and rock conditions have been similar to those of Mather "A," the helpful factors have been the hard metal tipped Carset bits and introduction of the hydraulic clamshell bucket for mucking.

The speed up and ease of mucking combined with the simplicity and low cost of the so-called hydro-mucker are important factors recommending expanding use. There seems to be little question that employment of this equipment should produce similar worth-while results in other sinking jobs no matter what their size or shape, the one evident requirement being that the shaft shall be vertical or nearly so.

Diamond Drilling Quartz-feldspar Intergrowths

By L. C. ARMSTRONG,* Member AIME

Twice in the past two years and in two widely separated localities—one near Williamsville, Mo., and the other in the Allard Lake district of Quebec—the Contract Drilling Division of the Longyear Company has experienced perplexing difficulties in drilling granitoid igneous rocks. The rocks in both places are characterized by appreciable amounts of feldspar intimately intergrown with quartz particles of microscopic dimensions. The disappointingly slow rate of advance and the exceptionally high bortz loss, obtaining during the life of these two projects, are ascribed largely to the resistance to abrasion of these two minerals when arranged in this intergrown pattern. Whether all intergrowth-bearing igneous rocks would offer extraordinary resistance to bit progress cannot be stated with assurance on the strength of these experiences alone. More data on diamond drilling this sort of material in a number of localities are needed before any generalizations can be made safely.

It should be pointed out that the lack of time and funds made it impossible for the writer, or other persons interested in correlating the details of bit histories with the kinds of rocks drilled, to be present during the performance of the contracts. However, the records kept in the normal course of drilling are deemed complete enough to be reportworthy. This information is presented here together with the findings made in studies of the rocks involved, with the thought that it may attract the comments of others who may have encountered similar refractory drilling conditions. A pooling of knowledge on the problem would be of material aid in devising more efficient cutting tools and procedures for penetrating this kind of rock.

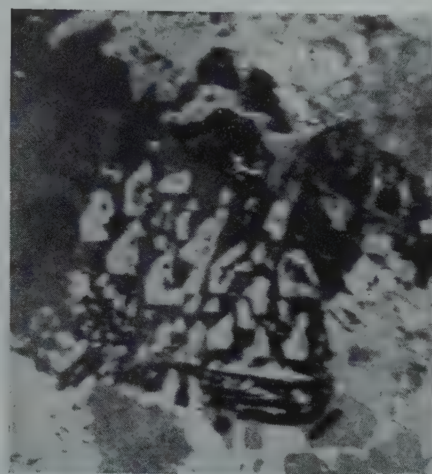

FIG 1—Micrograph of granite from Williamsville, Mo. showing graphic intergrowths. White areas are largely quartz; those with black and grayish hues are mostly feldspar (orthoclase and microcline). Crossed nicols. Approximate magnification: X 40. (Reduced approximately one fifth in reproduction.)

FIG 2—Micrograph of anorthosite from Allard Lake district, Quebec. Plumose areas with vermicular banding are myrmekite. Grains with straight parallel lines are labradorite. Crossed nicols. Approximate magnification: X 40. (Reduced approximately one fifth in reproduction.)

Drilling Blastholes in the Williamsville Granite

Due to the unsatisfactory performance of pneumatic percussion drills in boring blastholes on a construction contract near Williamsville, Mo., tests were made with a diamond drill. The poor progress achieved and the excessive loss of diamonds, however, soon led to the abandonment of these trials. The diamonds, in all of the bits tried, lost their cutting edges and acquired a polish after being in service for distances ranging from a few inches to a few feet. Coring bits containing surface-set, whole-stone bortz in a tungsten powder matrix were employed. The cost assignable to bortz wear averaged $7.17 per foot. This item of expense is several times the amount normally expended in diamond drilling in firm, unfractured, granitic rocks. Diamond-impregnated, tungsten carbide bits also were used without success. The diamond wear was at the rate of $0.98 per foot, but the diamond chips became too polished for further duty in less than a foot of drilling.

The Williamsville granite is pink and medium-grained. In the hand specimen, it has a fresh, sound appearance; it is seemingly composed almost entirely of interlocking grains of feldspar and quartz and has no discernible features suggesting a greater durability than is usually found in rocks with this composition and texture. Microscopic examination, however, reveals a predominance of feldspar grains with more or less regularly distributed and sharply angular masses of quartz (see Fig 1). The pattern assumed by the two minerals in this mode of occurrence is referred to as a "graphic intergrowth," as it bears some resemblance to the cuneiform characters of ancient Persian writing. The other principal constituents are quartz-free feldspars (microcline and orthoclase) and quartz. All of these constituents are in nearly equidimensional grains, ranging up to about 3 mm in length. There are also

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Table 1 . . . Composition of Granite from Williamsville, Mo., and of Anorthosite from Allard Lake District, Quebec

	Williamsville Granite (Estimated, Per Cent)	Allard Lake Anorthosite (Estimated, Per Cent)
Quartz-feldspar intergrowth	(graphic) 61.0	(myrmekite) 22.0
Feldspar	(potash) 21.0	(labradorite) 75.0
Quartz	15.5	
Total quartz and feldspar	97.5	97.0
Magnetite	1.0	0.9
Green biotite	0.5	
Green hornblende		0.2
Muscovite	0.3	
Augite		0.2
Chlorite	0.3	0.2
Epidote		0.2
Ilmenite and/or magnetite		0.2
Secondary minerals (probably largely epidote, chlorite, and calcite)		1.0

many finer-grained, less abundant accessory minerals and alteration products, including magnetite, green biotite, muscovite, chlorite, and leucoxene. The amount of each mineral present, estimated in a study of two thin sections, is listed in Table 1, and to facilitate comparisons, the components of the Allard Lake anorthosite are also given.

Drilling in the Allard Lake Anorthosite

According to accounts received from others,[a] prospecting in the Allard Lake district of Quebec for ilmenite entailed drilling in anorthosite and ilmenite ore, and in gradations between these two classes of material. Where the rock was dominantly ilmenite, the cost per foot and the drilling speed were about what would be expected in drilling in the Canadian Shield. Sections of anorthosite, on the other hand, were reflected by a slower rate of penetration and by an increase in the destruction of bortz. In anorthosite, a new bit would cut satisfactorily during the first few feet of drilling, but almost without fail the diamonds would acquire such a smooth polish in a 10-ft run as to make the bit unfit for further use in this kind of rock. Frequently, however, it was feasible and economically worth-while to do additional drilling in ilmenite ore with these worn bits before returning them for resetting.

Whole-stone, surface-set bits with a beryllium-copper matrix were supplied for the work. All were EXT, a Canadian bit size, which, except for a slightly larger inside diameter, has dimensions identical with the EX size used in the United States. No attempt was made while conducting the work to maintain separate data on the results obtained in anorthosite and in ore, but subsequently, those associated with the project have made fairly accurate estimates which indicate a cost of $2.00 a foot, or more, as being chargeable to bortz loss while coring anorthosite. This figure is considered very large in relation to bit performances in drilling the average pre-Cambrian rocks in Canada.

As represented by pieces of drill core, the anorthosite is a pinkish gray rock with a large percentage of coarse, plagioclase feldspar grains attaining lengths of approximately 1 cm. Some smaller, irregular-shaped, hard, gray masses occur scattered throughout the rock, but as they are not conspicuous, they would probably be overlooked in a casual inspection. Under the microscope, it may be observed that three-fourths of the rock is labradorite and that the hard, gray masses, occupying much of the remaining one-fourth, are composed of myrmekite, a vermicular intergrowth of quartz and feldspar (see Fig 2). The areas of myrmekite consist of thin "wormy" layers of quartz alternating with similar layers of feldspar. As this intergrowth replaces the labradorite, it was probably formed after the crystallization of the labradorite but before end-stage, deuteric action had ceased. Inconsequential amounts of other minerals, such as magnetite, augite, hornblende, epidote, chlorite, apatite, and calcite are also present. Some of the epidote and calcite, and possibly one or two other minerals, occur in thin reaction rims around grains of labradorite.

[a] Principally personal communications from P. A. Hermiston, manager of the Canadian Longyear, Limited, Contract Drilling Division.

Conclusions

There is little doubt that drilling the Williamsville granite and the Allard Lake anorthosite was affected adversely by the presence of quartz-feldspar intergrowths. On the basis of these two case histories, it is thought likely that rocks of this kind become progressively more unyielding as the content of quartz-feldspar intergrowth increases.

Feldspar is softer and more cleavable than quartz, and therefore would probably yield more readily to the shearing forces set up, when bortz are forced onto it and the bit rotated. For this reason, it is thought probable that a hole being drilled in rocks like those described above would be, at all times, a small fraction of an inch deeper in the feldspar than in the areas of quartz-feldspar intergrowth. In this somewhat elevated position, the relatively hard, tough intergrowths would be in an advantageous situation for exerting counter forces against the cutting points and edges of the bortz. As a result, the points and edges would be sheared off, and the bortz rapidly would become too smooth for further effective drilling. It is also more than likely that the bit would abrade the feldspar contained within the intergrowths slightly in advance of the associated quartz, and in doing this would continuously expose tiny, sharp bodies of quartz on the surface of the intergrowth. These small quartz projections, shaped somewhat like the teeth on a file, also would be instrumental in putting a fine polish on the surface of the bortz.

If the above conception of what takes place during the drilling of quartz-feldspar intergrowths is correct, the thought arises concerning the applicability to the problem of a bit made with very fine-grained diamond fragments in a suitable matrix. In using a bit of this type, the difference in relief between hard and soft constituents in the rocks being drilled would be almost nil, and furthermore, fresh, angular, diamond pieces would always be available for cutting until the bit was run to destruction.

Acknowledgments

Without the data, materials, and help given by members of the staff of the Longyear Company, the preparation of this paper would have been impossible. An expression of appreciation is due the Quebec Iron and Titanium Corporation for granting permission to publish information gathered during the drilling of their ilmenite property in Quebec, and also to the Winston Brothers of Minneapolis, Minn., for data pertaining to their activities near Williamsville, Mo.

Safety Practices at the Crestmore Mine of the Riverside Cement Company

By R. H. WIGHTMAN* and G. H. ADAMS,* Members AIME

In order to secure good results in the prevention of accidents, it is generally recognized that the desire for such accomplishment, as well as the aggressive and constructive action to achieve it, must emanate from the Top Management and permeate successively down through the "Chart of Authority" to the workmen.

Several years ago, with the above as a basis and with full support of the Top Management, the Mining Department started a program of selling the foremen and bosses the idea that safety could be achieved and that it was desired and required. Using psychology, persistence and numerous other methods, the foremen and bosses were finally impressed with the idea that "It can be done" and "It will be done." With this in mind, new ideas were brought forth by the foremen and bosses regarding the caving operation and the specific jobs in the operation. These new ideas were taken up in round-table discussions and resulted in certain changes in the undercutting method and the actual drawing of the blocks. These changes resulted in increasing fragmentation of the blocks which prevented the usual hang-up areas in the blocks. With the rock smaller in size and within reasonable reach from the tapping drifts, accidents were prevented in the tapping section. At the same time, an enforced campaign of housekeeping throughout the mine was undertaken with very beneficial results. The foremen and bosses then, in turn, educated and trained the workers themselves to become *safety conscious.* In so doing, a constant reminder was given on all jobs assigned that the job was to be done safely as well as efficiently. No man is assigned a job who is not qualified for the type of work required. Efficiency and safety go hand in hand.

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Safety Rules

To supplement and aid this *safety conscious attitude,* certain safety rules applicable to our operation, a modified block caving system of mining limestone, in addition to the established state regulations, were put into effect and enforced. The specific rules referred to above and which will be mentioned below are in most cases simply a means of applying and complying with the very adequate "Mine Safety Orders" issued by the Industrial Accident Commission of the State of California.

Some of the safety rules and policies are as follows:

All men are required to wear safety

shoes and hard hats. Certain jobs require the use of safety glasses. Safety glasses are available to all men upon request.

On the grizzly level each tapper must keep his working place clean and clear of broken rock. When blasting, the tapper turns on the blasting signal at the end of the drift where the blasting is to be done. The signal consists of a red light and an air whistle at the entrance to the drift. The signal also is connected with a howler on the haulage level, at a point directly below where the tapper is working, to warn the trainmen of blasting above. Tappers also turn on an air valve near the chute the tapper is working. The escaping air can be heard readily in the tapping section, and serves also as a warning and as an aid in clearing the area of smoke and dust.

Another requirement which is considered to be very important, in fact so important that noncompliance is cause for discharge, is that each tapper takes a dummy fuse with him and after spitting all his fuses for the blast he spits the dummy fuse. This dummy fuse must be equal in length to the longest fuse used in the blast. Counting the shots as they go off is difficult and even though the shots can be distinguished and counted the tapper cannot return to the drift until the dummy fuse which he has carried with him has burned out. We are confident that in many cases the use of the dummy fuse has prevented men from walking back into their own blast. The tappers also are careful to notify any other tapper nearby when they are going to blast. (Cooperation among the workmen themselves is a great factor for the prevention of accidents.)

Tappers and drillers are required to cover the grizzlies with lagging before starting to drill. High or dangerous chutes are inspected by the tapper boss before the tapping driller starts to drill. Buffer timber or rail is installed in exceptionally high or wide tapping throats. Periodic inspection of the back of drifts is made by the timberman or tapping blaster.

On the haulage level, one of two-man train crew always precedes the train when backing into a haulage drift. Trainmen are required to keep a portable and detachable red light on rear car of each train. Periodic inspection of draw sets is made by the timberman.

The trolley has automatic section switches. Wooden guards are placed over the trolley at draw chute positions. There is an air whistle on the tipple approach to warn trainmen to swing in the trolley arm for proper clearance before the motor passes into the tipple. When blasting at either the tipple or pocket, the operator must throw a switch which turns on red warning lights at both places.

Also, all foremen and bosses, as well as a number of key personnel, are trained in first aid work.

Other features of our safety program include a Mine Safety Committee composed of a supervisor and mine employees. This committee makes regular monthly inspections and submits recommendations for better safety. These recommendations are taken up by a plant committee and action within 30 days taken on those finally approved.

A similar committee makes regular inspections for "good housekeeping" only; and a report of their ratings as to poor, fair, good, and excellent is posted on the bulletin board. It is felt that the effect of the report has been beneficial in creating a competitive spirit among the individual employees in keeping their respective working places in proper condition—and good housekeeping is an integral part of safety.

A mine safety bulletin board is available on which safety posters are displayed, safety slogans are written and changed periodically; with the date of the last lost time accident, name of employee injured, and the number of days since the last lost time accident.

The company encourages safety by the incentives of a monthly pay bonus to each employee in those departments completing a month without a lost time accident; a safety dinner awarded to each department that completes a year without a lost time accident; and the award of a safety medal for each supervisor whose crew completes a calendar year without a lost time accident.

Other factors that have directly or indirectly helped us in our safety program are that our mine crew has been fairly stable and our "old timers" are thoroughly trained and qualified miners, are wholeheartedly for safety and will fight for it; and they are interested enough to help train and advise the occasional new men that are taken on.

In the final analysis, a successful safety program depends principally upon the experience, common sense, attitude, cooperation, and the entire support of all individuals making up the working force. The Management takes every opportunity to compliment the employees for their effort and gives them credit for the splendid record achieved.

Sintering Characteristics of Minus Sixty-five and Twenty Mesh Magnetite

By ALAN STANLEY,* Junior Member AIME, and JOSEPH C. MEAD

Introduction

The MacIntyre Development of the National Lead Co. is located at Tahawus, N. Y. The operations involve the mining and concentrating of a titaniferous iron ore to produce an ilmenite concentrate and a magnetite concentrate.

Construction of the MacIntyre plant was commenced during the summer of 1941, when world conditions threatened to cut off the supply of Indian ilmenite. An open pit mining operation was developed and the crushing and milling equipment put in operation in July 1942. A general description of the operation was given in the Adirondack Issue of *Mining and Metallurgy* for November 1943. The metallurgy of the mill operation was described by Mr. Frank R. Milliken,* Plant Manager, National Lead Company, MacIntyre Development, and presented at the AIME New York Meeting, February 1948.

The magnetite concentrate produced in the milling operation was too fine (minus 20 mesh) to be used directly in iron blast furnace operation, and most of the magnetite had to be stockpiled in 1942 and 1943.

In 1943, the Defense Plant Corp. built a Greenawalt sintering plant at Tahawus, N. Y., to put the magnetite concentrate in a more suitable form for use in the iron blast furnace.

The Greenawalt sintering plant consists of three 10 by 25 ft sintering pans designed to produce 1800 gross tons of sinter per 24 hr. The vacuum to each pan is produced by two Greenawalt fans in series, pulling approximately 30,000 cu ft of air per minute at 50 in. water gauge vacuum. The plant started operation in August 1944. The present plant production averages 25 tons per operating pan hour (approximately 224 lb per operating hour per square foot of grate area) of plus 1 in. sinter.

Raw feed to the plant consists of 61 pct magnetite, 4 pct anthracite coal culm, and 35 pct minus 1 in. return fines which are conveyed to a pug mill where the materials are mixed thoroughly and water added to give the mixture 5.5 to 6 pct moisture. The mixed prepared feed is conveyed to two 4 by 10 ft vibrating screens where the minus 1 in. plus ⅝ in. return fines are screened out and discharged into a surge bin for use as a hearth layer. The minus ⅝ in. prepared feed is discharged into another surge bin for use as prepared feed.

A charge car, electrically operated, having a capacity of one charge of prepared feed and several charges of hearth layer, lays a thin layer of plus ⅝ in. return fines and 9½ in. depth of prepared feed into the pans. A fluffing roll and a vibrator on the car fluffs and spreads the prepared feed into the pans.

An ignition car, electrically operated, ignites the top of the bed with a 30 sec flash burn. The 9½ in. bed sinters in approximately 13 min. Dumping the pan, and recharging and igniting the bed requires 2 min.

To improve the quality of the ilmenite concentrate produced in the mill and to reduce the amount of titanium dioxide lost in the mill tailings and in the magnetite product, extensive research work and pilot plant operations have been done on grinding the crude ore to minus 65 mesh size (rather than to minus 20 mesh) and concentrating it by a combination of magnetic separation (for magnetite recovery) and flotation (for ilmenite recovery). These tests have proved successful in increasing ilmenite recovery and grade.

With the development of the ilmenite flotation process to a stage where a full scale flotation plant was in the design stage, the problem arose of handling the 65 mesh magnetite concentrate that would be produced. In order to study and solve the problems of handling and sintering the 65 mesh magnetite in the sinter plant, a pilot sinter plant was secured from John E. Greenawalt.

The effect of using 65 mesh magnetite in the sintering operations was then studied on the 2.4 sq ft test pan, operating under conditions as similar to the large plant as could be set up in the laboratory.

A series of tests were run in the test pan on present sinter plant feed that had been mixed in the plant pug mill. An average production and an average quality of sinter produced in this series

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* Frank R. Milliken: Metallurgy at National Lead Company. TP 2355, AIME *Mining Tech.* (May 1948).

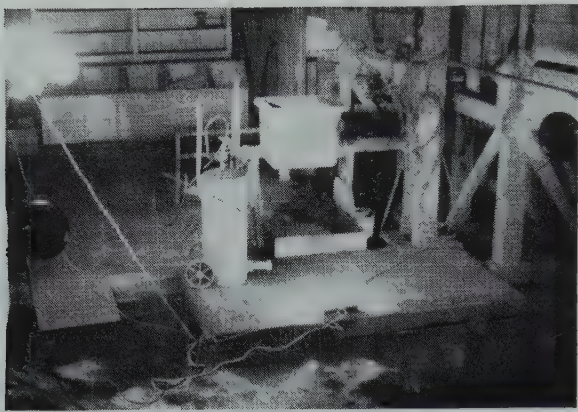

FIG 1—Sinter pan and kerosene burner.

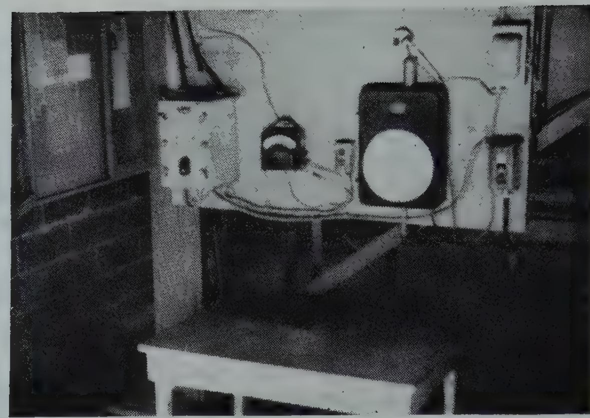

FIG 2—Instrument panel.

of tests was established as a "Standard" (8.8 lb per min) to compare the sintering of minus 20 mesh magnetite with that of minus 65 mesh magnetite.

The high points on each series of tests were repeated twice as a check on the original, to be sure that some unknown variable did not cause the change in the sintering performance. A test run of 10 pans was made on the mixture found to give the best production and quality.

Procedure

The pilot sinter plant included a sintering pan 20¾ by 16¾ in. and 9 in. deep, an ignition hood, a pressure kerosene burner for ignition, a dust collector, and a 400 cfm vacuum fan rated at 54 in. of water (see Fig 1 and 2 for instrument panel, sinter pan and kerosene burner).

Vacuum was recorded on a Bristol meter, and exhaust temperatures were read from an electrical pyrometer.

Magnetite of 65 mesh was prepared by grinding stockpile (20 mesh) magnetite, to the desired size, in a batch ball mill, and filtering in a laboratory pressure filter, Table 1.

The moisture obtained by this method of filtering was too high for pilot plant use (averaging about 9 pct) and it was necessary to dry a portion of the magnetite prior to its use in the sintering operation.

For the final "production run" tests, rod mill screen undersize (20 mesh) was ground to 65 mesh, and the magnetite was concentrated in a laboratory Crockett.

Coal was obtained from the sinter plant stockpile, and when necessary, ground in a ball mill to the desired size. After several tests had to be discarded because of poor results caused by coal with high ash content, all coal samples were submitted to the laboratory for ash determination. In order to eliminate one variable, only coal with a 10 pct or less ash content was used in the test work.

Sawdust, when needed, was obtained from local mills; limestone and slaked lime were obtained from the Chazy Lime and Stone Co.

The return fines, unless otherwise noted, were obtained by crushing plus 1 in. sinter, produced in the pilot plant, in a jaw crusher and rolls and screening it to the desired size on a laboratory vibrating screen.

Screens for screening the sinter were made locally, as were sample racks and storage bins for the raw materials. A ¼ yard gasoline concrete mixer was connected to an electric drive and used for the mixing of all batches. Balances and a hot plate were provided for moisture determinations, and an electric clock with a sweep second hand was used for timing the tests.

A group of five persons was found to be the most efficient, as continuous test work required constant work by one man grinding magnetite and coal, one man preparing sized return fines, two men preparing batches for burning, and one man completing the necessary calculations and paper work. The last three men were required when the batches were burned.

Table 1 . . . Screen Analysis of Magnetite Products

	+28	+35	+48	+65	+100	+150	+200	−200
20 mesh	8.3	16.4	19.5	14.8	12.1	9.2	5.2	14.5
65 mesh				3.0	11.2	19.8	15.8	50.2

Note: All references to 65 and 20 mesh magnetite refer to minus 65 and minus 20 mesh material.

Each batch mixed contained enough material for two tests. The charge was calculated in advance. A weighted mixture of dry and filtered magnetite was added to the mixer. This material was then mixed for 5 min and a moisture determination made. Using this moisture it was possible to calculate the exact amount of magnetite that had to be removed or added to the mixer to leave the required weight of magnetite at 5 pct moisture.

Then the remaining materials were added dry, and the necessary amount of water to provide the required moisture was sprinkled, with a sprinkling can, into the revolving mixer. When trouble with pellet formation occurred, the procedure was changed so that the necessary water was added to the return fines, and then the wet returns were added to the mixer. The latter method was more satisfactory.

Each batch, unless otherwise noted, was mixed for 5 min again.

A 15 lb bedding charge (minus 1 in. plus ½ in. sinter) was added to the pan for each burn. This gave a bedding depth of 1 in. Then the sinter pan was filled, with a weighed amount of mix, and dumped by hand through a 1 in. screen. The bed then was made level by scraping off the excess material. The weight of the mix in the pan was obtained by difference. The ignition hood was lowered into place over the pan, and the pressure kerosene burner was ignited, as the vacuum was turned on, and ignition took place for 30 sec. The hood was then raised. A record was made of the exhaust temperature as the burning proceeded and vacuum was recorded continuously with the Bristol meter. When the bedding on the grate bars glowed red (viewed by means of a mirror mounted to make the grates visible through a hole covered with glass in the bottom of the pan) the vacuum was turned off and the

sintered charge was dumped by inverting the pan.

The results of the first pan of the mix were usually poor because a cold pan did not approximate actual operating conditions. The second, or hot pan, gave results that could be compared more directly with sinter plant practice.

After each pan was burned, the material was passed over a 1 in. Ty-Rock screen three times. The material remaining as plus 1 in. was "sinter." The minus 1 in. material was passed over a ½ in. Ty-Rock screen twice, and the plus ½ in. material was "bedding." The remainder of the material (minus ½ in.) was "return fines" (note: in all references to return fines in this paper, the bedding is excluded).

In order to have a strength comparison of the sinter formed by various mixes, a "quality" figure was initiated. This figure was determined by screening the cold (the plus 1 in. sinter was allowed to air cool overnight) sinter in the same manner as above. The weights of the plus 1 in. and the plus ½ in. were combined, and this total weight was divided by the weight of the hot 1 in. sinter to obtain relative "quality."

Cycle time was figured by adding 2½ min to the burning time to allow for dumping and recharging the pan. Theoretical production in pounds per minute was determined by dividing the pounds of plus 1 in. sinter by the cycle time. In quite a few tests the amount of bedding and return fines produced was not sufficient for repeat operations. Because of this some of the plus 1 in. sinter had to be taken to provide the amounts of these materials needed. Actual production was then the weight of plus 1 in. sinter remaining after bedding and return fines requirements were met, divided by cycle time.

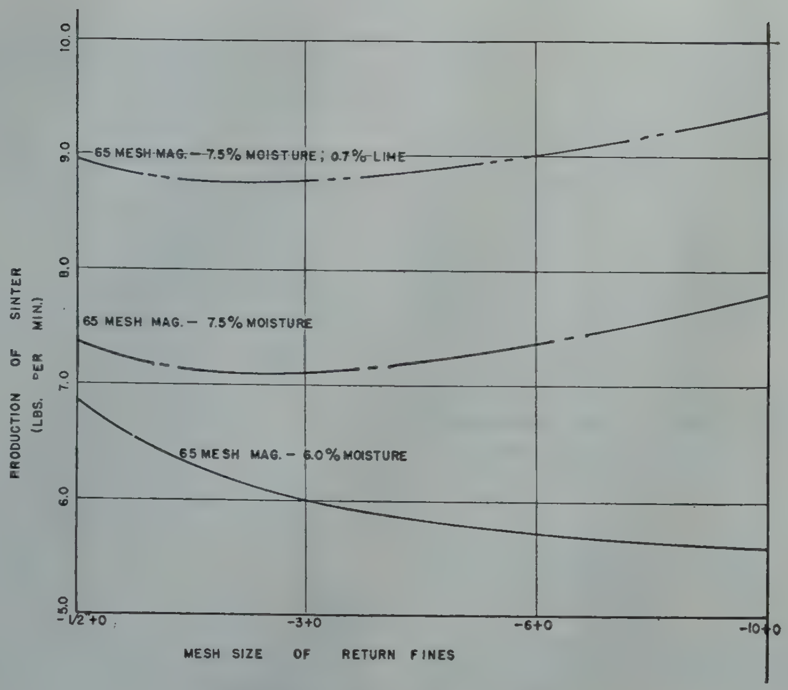

FIG 3—Results of return fines series.

Discussion of Factors Investigated

The following factors were investigated during the test period:

(1) return fines, (2) coal, (3) moisture, (4) slaked lime and miscellaneous reagents, (5) oxidation, (6) limestone, (7) sawdust, (8) pellets, (9) "dust," (10) mixing time, and (11) production runs.

RETURN FINES

As far as is known, there is no sinter plant in the United States that employs sized return fines as a necessary metallurgical requirement for its operation. Mr. Worm Lund, on a visit to MacIntyre from Sweden, suggested the possibility of sized return fines in the sintering of 65 mesh magnetite.

In the test plant at 20 mesh, return fines were not necessary for the production of good sinter. In many cases the return fines were detrimental to test plant production. This is especially true when the returns consisted mainly of raw magnetite, coal, and dust from the dust collectors.

For sintering 65 mesh and finer magnetite, return fines are absolutely necessary for sinter production. If efficient operation and economical sinter are to be had, it is necessary to have these return fines fall within definite size ranges. The secret of sintering fine concentrates is to obtain proper porosity in the "bed" so that air can be uniformly distributed throughout and burning can progress.

A series of tests were run using minus ½ in. plus 3; minus 3 plus 6, minus 6 plus 10, minus 10 plus 20, minus 10 plus 35, and minus 20 plus 35 mesh return fines. The best results were obtained from the minus 10 plus 35 mesh tests. Because of the difficulty that would be incurred by screening to these close sizes, a series was run using minus ½ in. plus 0, minus 3 plus 0, minus 6 plus 0, and minus 10 plus 0. The results of this series can be seen in Fig 3. These tests were run with 4.0 pct coal and 25 pct return fines.

As can be seen (Fig 3) the best results were obtained using minus 10 mesh plus 0 return fines. The best percentage of this size return fines was determined by a series of tests (Fig 4). The results were satisfactory only as long as the minus 35 mesh portion did not exceed 25 pct of the total return fines weight. (This then would give a calculated minus 10 plus 35 mesh return fines figure of 18.75 pct of the total weight of the charge.)

A study of the results obtained using various amounts and sizes of return fines indicated:

1. Proper porosity is essential in producing sinter economically:
   a. Too much porosity causes too rapid burning and results in poor sinter.
   b. Insufficient porosity increases the burning time beyond economic limits.
2. Excess returns weaken the sinter and cause a drop in production.
3. By proper control of the size and amount of returns it should be possible to sinter magnetite finer than 65 mesh.
4. More efficient sinter plant operation could be obtained if the dust from the dust collectors and the finer portion of the returns were returned to the original source of sinter plant feed (for example: to magnetite filter).

A 20 mesh magnetite will sinter without the addition of return fines, providing adequate bedding is supplied, but blast furnace operations object to fines; this requires that the fine material be screened out and recirculated.

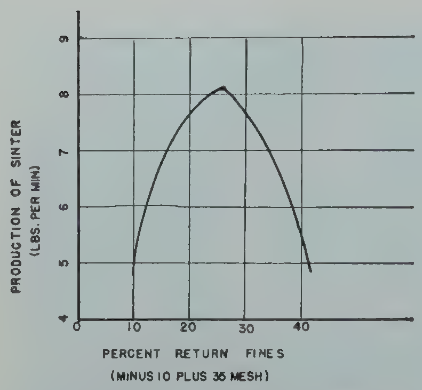

FIG 4—Results of various percentages of minus 10 plus 35 mesh returns.

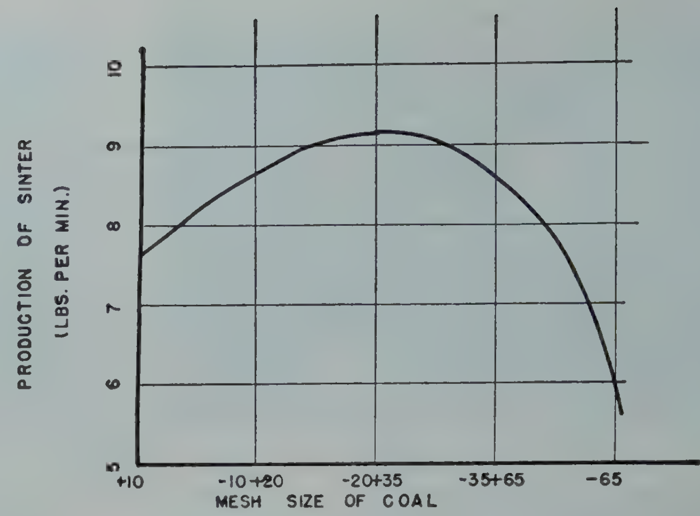

FIG 6—Effect of various mesh sizes of coal on sinter production using 65 mesh magnetite.

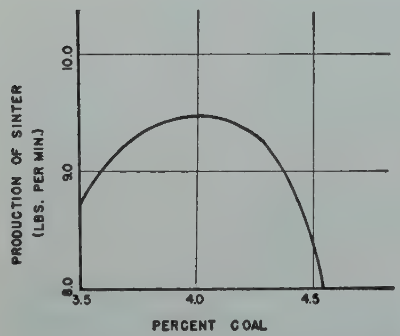

FIG 5—Effect of various percentages of anthracite culm 4 and 5 buckwheat coal on sinter production using 65 mesh magnetite.

COAL

Coal is used in MacIntyre sintering as the most economical local source of carbon.

The amount and size of coal used is an important factor in a sintering operation.

As can be seen in Fig 5, 4.0 pct coal produced the best results in sintering 65 mesh magnetite. Both greater and lesser amounts of coal tended to slow the burning time and reduce the production of sinter. Insufficient coal did not produce enough heat for proper sintering and excess coal caused too high a temperature, resulting in slagging and overoxidation with resultant poor sinter. This overoxidation was caused by an oxidizing atmosphere, and while reduction may have taken place during the initial phases of the burning, the final result was excess oxidation.

On the theory that coal used in sintering should be approximately the same particle size as the magnetite, the coal for the sintering tests was ground to 65 mesh. This fine coal, at first, seemed to help sinter production, but sawdust was being used in conjunction with it. As testing progressed, satisfactory results were not obtained, and it was decided to test various sizes of coal. These tests (Fig 6) proved that fine coal, rather than help, did much to deter good sinter production. The No. 4 and 5 buckwheat anthracite culm, as received for 20 mesh sintering, gave excellent results and made it possible to eliminate sawdust from the mix.

The fine coal tended to plug most voids, and made proper air flow impossible. The coarser coal left large voids as it burned, causing lower vacuum and faster burning time.

MOISTURE

The amount of water that can be added to a 65 mesh magnetite "mix" is limited. The "mix" will sinter, if all other proportions are correct, at a moisture between 6 and 8 pct, with the best results obtained at 7.5 pct moisture (Fig 7).

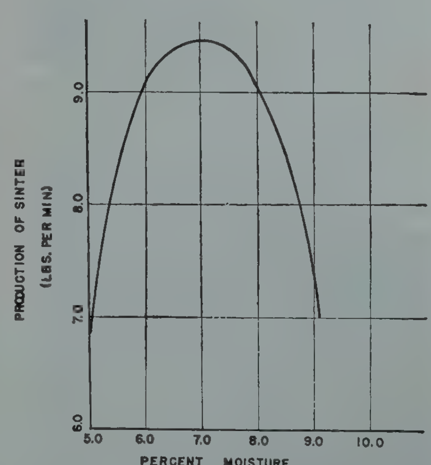

FIG 7—Effect of various percentages of moisture on sinter production with 65 mesh magnetite.

If the moisture is too high, the mix becomes semifluid and it is impossible to draw air through the bed. Thus, ignition is impossible. If the moisture is low, holes tend to form as the material is pulled through the grates. The air, following the path of least resistance, is drawn through the holes and thereby short-circuits the bed. Poor burning results and less sinter is produced.

With 20 mesh magnetite, a wider moisture range is possible. Good sinter was produced with a moisture as low as 3 pct, and as high as 8 pct. However, with 20 mesh magnetite, the percentage of moisture required is closely related to the amount of dust present in the returns. If there were more than 35 pct of minus 35 mesh material in the return fines, it was necessary to use a moisture of 7.5 pct (production with the same mix at 7.5 pct moisture was 10.8 lb per min and at 6.0 pct moisture, production was 9.6 lb per min). If the returns contain less than 35 pct minus 35 mesh material, best results are obtained with a 6 pct moisture.

SLAKED LIME AND MISCELLANEOUS REAGENTS

In an effort to reduce burning time, and thus increase production, slaked lime was added to the charge. A 10 pct increase in production was obtained with the addition of 0.7 pct lime to the 65 mesh mix (Fig 8).

If an excess of lime is added, the charge will burn hot and fast, with little sinter being produced. It is thought that the lime causes an increase in the surface tension of the water. With this higher surface tension, the water would fill less voids in the

Table 2 . . . Effect of Lime on Varying Amounts of Coal

	Production (Lb per Min)	Cycle Time (Min)	Temperature Exhaust Gas Degree C	Quality	Vacuum In. of Water		Per Cent Lime
					Start	End	
With 3 pct coal...	5.5	9.0	520	81.5	61	48	0.75
With 4 pct coal...	11.0	10.0	325	82.2	57	53	0.75

Table 3 . . . Effects of Various Reagents on 65 Mesh Magnetite Sinter

Reagent	Per Cent Reagent	Production, Lb per Min	Per Cent Plus 1 In. Sinter	Quality	Cycle Time, (Min)
Standard		9.0	65.1	78.9	13.1
Lime	0.7	9.9	63.6	78.0	11.5
Detergent	0.005	8.1	64.2	86.0	15.0
Oleic acid	0.05	(would not burn)			
Calcium chloride	2.0	9.7	61.4	74.8	12.0
Pine oil	0.05	5.1	52.2	88.2	20.5
Zinc stearate	0.5	(would not burn)			
Aluminum dust	0.5	5.8	54.8	78.9	18.0

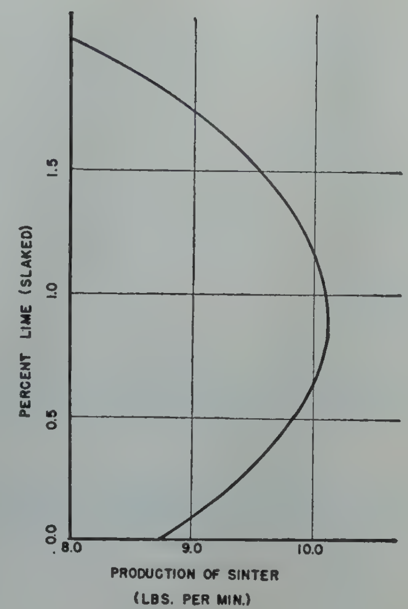

FIG 8—Effect of various percentages of slaked lime on sinter production with 65 mesh magnetite.

mix, thus leaving greater and more numerous voids than possible with unadjusted water. With more numerous voids, a greater volume of air was drawn through the bed, lowering the vacuum, decreasing the burning time, and causing a higher burning temperature. The addition of excess lime increased the surface tension to such an extent that an excess volume of air was drawn through the charge, causing such rapid and hot burning that little sinter was produced.

If the percentage of lime was held constant and the coal percentage dropped, less sinter was formed because of higher burning temperatures. Higher burning temperatures slag the sinter.

The results of Table 2 show that with 3 pct coal and 0.75 pct slaked lime, the mix burned hotter, and gave poorer results than those obtained using 4 pct coal and 0.75 pct slaked lime. This can be explained by the knowledge that a higher percentage of carbon causes excess slagging. This slagging reduces the amount of voids, and this tends to slow the burning rate. This shows, again, how critical the addition of lime would be in actual plant practice. If lime were added in excess, production would drop. If the amount of lime remained constant and the percentage of carbon dropped, production would again drop. Thus, if lime is used in actual plant practice, efficient metallurgical and quantity controls must be provided.

In order to check the surface tension theory, a series of tests were made using the following reagents (Table 3): (1) detergent ("Dreft"), (2) oleic acid, (3) calcium chloride, (4) pine oil, (5) zinc stearate, and (6) aluminum dust.

It is known that a detergent reduces the surface tension of water. Thus, if a detergent is added to the mix, a poorer sinter should result, because with a lower surface tension, the water would fill more voids. The results obtained by test work checked with this idea. When 50 g of detergent were added to the mix, the mix became "putty-like" in appearance and semifluid in nature. The sized return fines were completely disintegrated when they were placed in water containing a detergent. When this mix was added to the pan it was impossible to draw air through it, and thus it could not be ignited.

Calcium chloride, which gives a substantial increase in the surface tension of water, was next added to the mix. The calcium chloride decreased the burning time, but produced a weak sinter. The sinter cake actually expanded during burning when calcium chloride was in the mix. Why this expansion took place is unknown. Calcium chloride could not be used in sinter plant operation because of the large amounts of chlorine gas liberated during the burning; however, it was used in the pilot plant because it was known to increase the surface tension of water.

All other reagents tried gave poor results.

OXIDATION

For a short while during the test work, it was thought that the oxidation of the FeO to Fe_2O_3 was an important factor in strong sinter. To check this point many assays were made on magnetite feed, and sinter produced from this magnetite. As a result of a study of these assays it is thought that oxidation to a certain extent does produce stronger sinter by causing the iron molecules to share oxygen atoms. When oxidation progresses too far, the iron is oxidized completely and no longer shares oxygen atoms, thus producing weaker sinter. When oxidation is low, there is little sharing of oxygen atoms and again the sinter is weak.

Table 4 . . . Effects of Oxidation upon Quality of Sinter

Magnetite			Sinter			Quality
Fe	FeO	Fe_2O_3	Fe	FeO	Fe_2O_3	
57.0	31.04	46.99	56.8	21.78	57.06	75.8
56.7	31.61	45.91	56.0	16.52	61.68	87.8
58.0	32.33	46.97	56.5	9.63	70.05	72.8

Table 4 is representative of the average assays completed, and shows weaker sinter on both the high and low oxidation samples.

LIMESTONE

Because of the cheaper costs of limestone compared to slaked lime, it was thought desirable to check the effects of limestone in the "mix." The limestone was sized from minus 6 plus 10, minus 10 plus 20, minus 20 plus 35, and minus 35 plus 65 mesh. In no case did the limestone help. This was probably caused by the fact that the heat necessary to slake the lime evaporated all the moisture before the limestone was slaked, and thus the effect of the slaked lime on the water was lost.

SAWDUST

Sawdust was helpful when 65 mesh coal was used in the mix. The sawdust

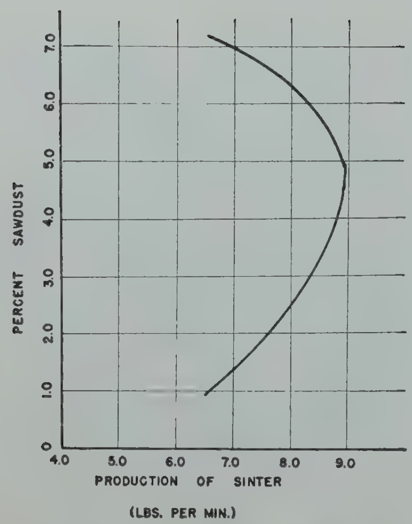

FIG 9—Effect of various percentages of sawdust on sinter production with 65 mesh magnetite.

had a tendency to "fluff" the mix in the pan, and it also made voids for air passage as the mix was burned. Fig 9 shows a high point for sawdust addition. As the percentage of sawdust was increased above 5, the mix burned very hot and fast and sinter production dropped. Sawdust elimination simplified the general problem because of the difficulties that its handling would have entailed if it were needed for 65 mesh magnetite sintering. The use of No. 4 and 5 buckwheat coal eliminated the need for sawdust.

PELLETS

A rotary mixer was used for pilot plant mixing. During the mixing of some batches, pellets were formed by magnetite building up around particles of return fines. The coarser the return fines, the larger the pellets would become. These pellets interferred with sinter production by causing short-circuiting of air around them because they had no porosity. Therefore they were unable to be sintered. The pellets were baked by the heat around them, but crumbled when subjected to the slightest pressure. No figures were obtained and the only method of making these determinations was by visible observation.

In sinter plant operation, pug mills are used for mixing and this problem should not occur. However, if pellets do form, some method to eliminate them must be provided. This does not infer sintering of a completely pelletized charge was undertaken. In all cases the pellets were mixed with unpelletized material, and under such conditions, were detrimental.

DUST

Dust is defined in this test work as material in the return fines, less than 65 mesh for 20 mesh magnetite sinter and minus 200 mesh for 65 mesh magnetite sinter. After it was known that excess fine material would reduce sinter production with 65 mesh material, tests were made to determine what effects "dust" would have on minus 20 mesh material. It was found that if the moisture was increased as the percentage of "dust" increased, the amount of "dust" in the "mix" had little effect on sintering of 20 mesh magnetite.

MIXING

Early in the project, several mixing tests were run using pug mill discharge. When this material was further mixed in the rotary mixer, a sharp production increase was noted. This mixing was checked again at the end of the project, and no increase in production could be attained by further mixing. In some cases production fell off after additional mixing, probably caused by water filling the voids in the return fines. It is thought that the pug mill blades, and other plant conditions were responsible for the poor results obtained during the first testing, but with the pug mills in good shape, as far as blade wear is concerned, a complete mix should be attainable. It was found that a 5 min rotary mix, after the magnetite had been brought to the correct moisture, gave the best results for rotary mixing in the test plant.

PRODUCTION RUNS

When the test work was completed it was necessary to test the best results. For this purpose, several pans were burned in rapid succession to see how the "mix" would react in a hot pan over a period of time. The recommendations made were on the basis of the production run results.

Conclusions

The best quality and quantity of sinter on this test pan were obtained with the following mix:

4.0 pct coal
25.0 pct return fines (minus 10 mesh plus 0) (at least 18.75 pct minus 10 mesh plus 35 mesh)
70.3 pct magnetite (minus 65 mesh size)
0.7 pct slaked lime.

The average production on the test pan with the above "mix" was 9.9 lb per min or 246 lb per hr per sq ft of grate area. This is slightly higher than average production in the sintering plant pans on minus 20 mesh magnetite.

The pilot plant work indicates that minus 65 magnetite can be sintered in the sinter plant if close control can be maintained over the following items:

1. Adequate amount of bedding material (minus 1 in. plus ½ in. return fines for hearth layer.
2. Sized return fines (minus 10 mesh plus 0).
3. Anthracite coal (size and moisture percentage).
4. Moisture percentage in the "mix."
5. Magnetite (moisture percentage).
6. Percentage of magnetite, coal, lime, and return fines in the prepared feed to the plant.

The results obtained in the sinter testing show that a more accurate metallurgical control on overall operation will be necessary for sintering minus 65 mesh magnetite than is needed in present plant operation for sintering minus 20 mesh magnetite.

Before the present sintering plant can handle minus 65 mesh magnetite, the plant flowsheet will have to be changed to include the conveying, cooling, screening, storing, and weighing of both return fines and bedding material.

Because of the difference between the size of the test pan (2.4 sq ft grate area) further development work will be needed in the sintering plant. A production run of at least two months would be required on minus 65 mesh magnetite to study the effect of the finer mesh size magnetite on the operation and maintenance of the plant.

Acknowledgment

The authors wish to thank the following individuals for their suggestions and help in making the sintering tests and for their help in making it possible for them to write this paper: Messrs. Gloyd M. Wiles, Chairman of the Mining Committee, National Lead Co., Frank R. Milliken, Plant Manager, MacIntyre Development, National Lead Co., A. R. Reiser, former Assistant Plant Manager, M. F. Morgan, Arthur G. McKee & Co., John E. Greenawalt, and the following metallurgical personnel of MacIntyre Development; Messrs. Ernest M. Spokes and Elton Geist.

Humphreys Spiral Concentration on Mesabi Range Ores

By WHITMAN E. BROWN,* Junior Member AIME, and LOUIS J. ERCK†

The installation in 1948 of a Humphreys spiral concentrator section at the Hill-Trumbull plant of The Cleveland-Cliffs Iron Co. is the latest commercial method on the Mesabi Range being used for the recovery of fine iron ore. In two stages of concentration 84 spirals treat approximately 120 long tons per hour of minus ⅛ in. ore. These spirals augment the production of the heavy-density plant which recovers the plus ⅛ in. iron from the plant crude ore. Structure of the ore is such that, when crushed, about 50 pct of the plant crude is minus ⅛ in. size. After desliming in a 66 in. Akins classifier, grinding the classifier product in a ball mill and again desliming in a 78 in. Akins classifier about 15 pct of the crude ore remains to be treated in the spiral plant.

Overall recovery of the ball mill feed in the Hill-Trumbull spiral plant during the 1948 season was about 53 pct of the weight and 66 pct of the total iron content. Of the actual feed delivered to the spirals from the 78 in. classifier 64.9 pct of the weight was recovered containing 73.7 pct of the total iron. The average analysis of the spiral concentrate is 55.05 pct Fe and 14.83 pct SiO_2.

An average of mill assays during the 1948 season is given in Table 1.

Table 1 . . . Average of Mill Assays during 1948

Product	Assays, Pct	
	Fe	SiO_2
Classifier product (ball mill feed)	44.20	31.47
Spiral feed (deslimed ball mill product)	48.50	24.84
Spiral concentrate	55.05	14.83
Spiral tailing	36.39	43.01

The Hill-Trumbull plants treat both wash ore and jig-type ore. During the 1948 season spirals were used only on the fine sizes of jig-type ore. At times the deslimed minus ⅛ in. size is of sufficient grade to warrant bypassing the spirals but in the main the fine size is delivered to the 48 first stage spirals and thence to the 36 cleaner or second stage spirals.

In the past several years numerous Mesabi Range ores in the fine size category have been tested by a wide variety of beneficiating equipment. The prodigious amount of research devoted to the recovery of values in the fine ore reflects the dire need of the iron industry to squeeze the last possible unit of iron from the mined ore and to develop suitable equipment and processes to satisfactorily concentrate the leaner horizons as they are encountered in the process of mining. Large governmental and private appropriations have been allotted, new laboratories built, research and technical staffs increased, and operator-manufacturer cooperative agreements applied to hasten the solution of the problem.

The University of Minnesota, at its Eighth Annual Mining Symposium in January 1947, dedicated as its objective the bringing to light of technological developments in iron ore beneficiation. Papers were given at that time relating to processes being developed. Specifically, these papers reviewed flotation, Baum jig, selective media concentrator, Hardinge concentrator, Dorrco sizer, heavy media concentration, Hydrotator, and the Humphreys spiral.

The Cleveland-Cliffs Iron Co., after encouraging results from tests in a small pilot plant using one full size spiral, made the first commercial installation of Humphreys spirals on the Mesabi Range at their Hill-Trumbull mine, Taconite, Minn. This company

San Francisco Meeting, February 1949.

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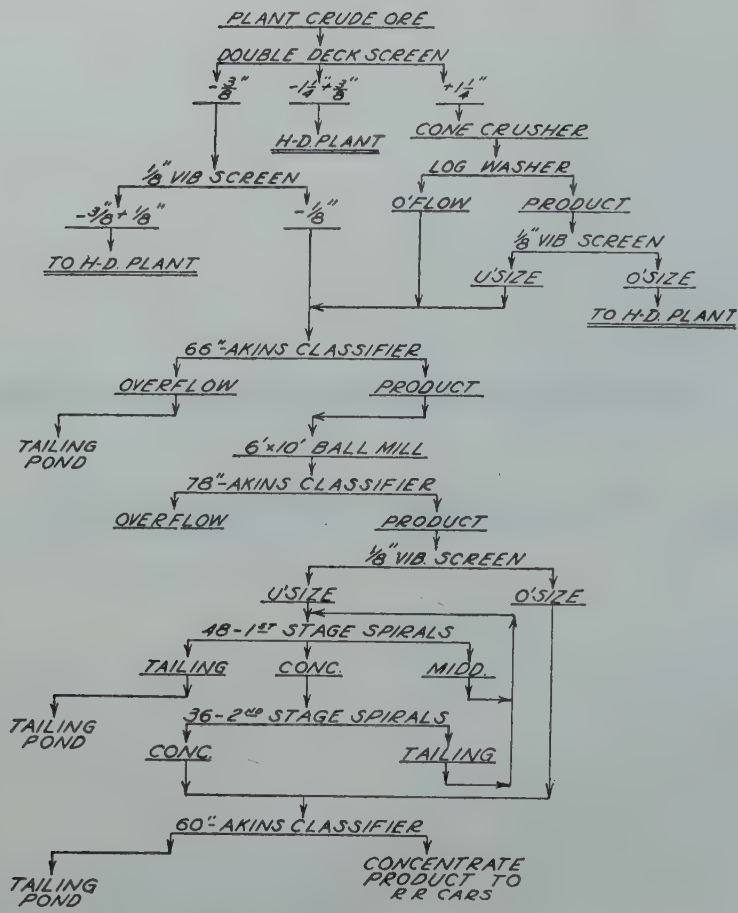

FIG 1—Flowsheet of fine ore section Hill-Trumbull plant.

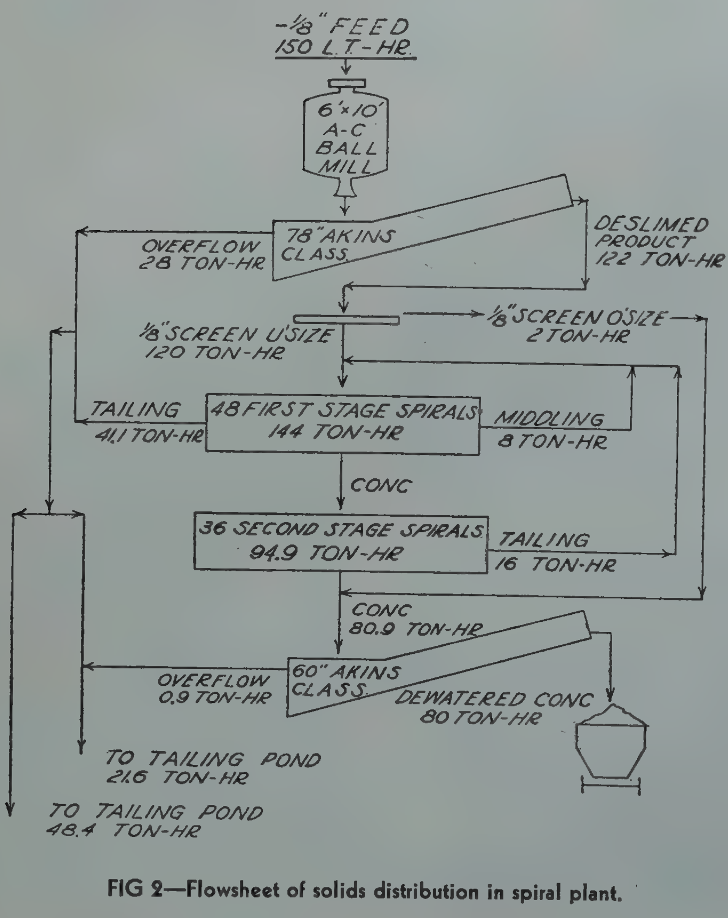

FIG 2—Flowsheet of solids distribution in spiral plant.

had tried in the laboratory and commercially, several other methods of concentration on the fine ores representative of the Hill-Trumbull group. These other methods of concentration failed to produce consistently an acceptable grade and recovery of finished product. Such failure was largely because of the inability to reject silica and interfering middling in the coarser sizes. This is particularly true of any method of concentration, either free-settling or hindered, in which a vertical component is used for eliminating the middling fraction. Success of the spiral is partly because of the centrifugal force with a horizontal component, resulting in substantial elimination of the coarser middling and tailing fractions while treating a wide range of sizes.

The Cleveland-Cliffs Iron Company Plant

A general flowsheet of the plant treating the fine sizes of Hill-Trumbull jig-type ore is shown in Fig 1. Flowsheet of solids distribution in the spiral plant is shown in Fig 2. Preparation of ore for the heavy-density plant is near standard practice. A slight divergence begins where the minus ⅛ in. size material leaves the 66 in. Akins classifier as a product and is routed to a 6 by 10 ft Allis-Chalmers ball mill. The ball mill operates in open circuit, is charged with 2 in. diam balls, driven by a 200 hp motor and has a throughput of approximately 150 long tons of solids per hour. Pulp density in the mill is approximately 65 pct solids. The purpose of the ball mill is to subject the ore to a differential grind which liberates silica from the iron ore particles, and middling from the ore particles and from silica. It also assists in breaking down porous iron oxide particles. The ground product flows into a 78 in. Akins classifier which makes a size separation at nearly 200 mesh. The slime or overflow waste analyzes about 25 to 30 pct Fe and 50 to 60 pct SiO_2. Classifier product containing about 75 pct solids drops onto a 4 by 8 ft. Allis-Chalmers Low-Head vibrating screen. The screen cloth has ⅛ by 2¾ in. slotted openings and is set with the long dimension of the slot parallel to the flow of the pulp. Experience has proved that as a general rule the plus ⅛ in. particles, after the abrasion grind, are of acceptable grade for final plant concentrates. Average analysis of the screen oversize was 54.20 pct Fe and 15.24 pct SiO_2. Primarily, the pur-

Table 2 . . . Power Distribution in Spiral Plant

Item	Connected Horsepower	Hp-hr per Ton of Feed to Ball Mill
Ball mill (6 by 10 ft)	200	1.333
78-in. Akins classifier (desliming ball mill product)	20	0.133
Vibrating screen (4 by 8 ft)	5	0.033
First stage feed pump (8 by 6 in.)	40	0.266
Second stage feed pump (8 by 6 in.)	25	0.165
Concentrate pump (5 by 4 in.)	30	0.198
Tailing pump (10 by 8 in.)	50	0.333
60-in. Akins classifier (dewatering concentrate)	7.5	0.050
84 Humphreys spirals	0	0.000
Total	377.5	2.511

Akins classifiers by Colorado Iron Works Co. Denver, Colo.

Ball mill, screen and pumps by Allis-Chalmers Manufacturing Co., Milwaukee, Wis.

The largest single power requirement is charged against the ball mill but preliminary spiral test results indicated a differential grind was justified for good recovery and separation.

pose of the screen was for removal of tramp oversize to prevent plugging pipelines and ports in the spiral installation. Usually the amount of screen oversize was a small percentage of the total feed and did not influence seriously the grade of the final concentrate.

Screen undersize flows to a sump which collects feed for the 48 first stage spirals. These spirals produce a finished tailing, middling, and a rougher concentrate. Middling is returned to the first stage feed sump. Rougher concentrate drops into another sump which collects the feed for the 36 second stage spirals. The second stage spirals produce a finished concentrate and a tailing. This tailing is returned to the first stage feed sump. A circulating middling is thereby accumulated in the rougher spirals consisting of the first stage middling and the second stage tailing amounting to about 15 pct of the new feed. Finished spiral concentrate joins the ⅛ in. screen oversize and is pumped to a 60 in. Akins classifier for dewatering prior to dumping into railroad cars. The heavy-density plant concentrates are dumped directly into the railroad cars.

Spiral Plant Operation and Performance

POWER

Table 2 shows distribution of power requirements associated with this spiral installation.

As operated during the 1948 season and based on the 150 long tons of feed per hour, power requirements for material handling in the spiral section were 2.51 hp-hr per ton of concentrate. This is equivalent to 0.40 tons new feed per horsepower-hour and 0.21 tons finished concentrate per horsepower-hour. The only power used in the actual concentration is in the 78 in. Akins classifier; the remaining power is used in grinding and material handling.

LABOR

All 84 spirals are policed by one operator who removes extraneous matter from the spirals, adjusts pulp density of spiral feed, and adjusts splitters and quantity of wash water. He also has time for additional duties including sampling and tending to the ball mill, pumps, classifiers, and screen.

If there is a power failure the spirals empty themselves by gravity.

Spiral operators are easily trained and after a reasonable length of time can become proficient. They are required to observe structure of the ore and to note changes which would suddenly increase or decrease the pulp density. When such changes occur, water is added or decreased to maintain the proper pulp density and launder make-up water. Occasionally, the Hill-Trumbull plant crude ore does not provide a sufficient amount of minus ⅛ in. material to adequately feed the spiral section. Before ringing for more feed the spiral operator consults the heavy-density plant foreman to determine how much more feed his section can stand. If the heavy-density plant is operating at maximum capacity and there is not enough ore

Table 3 . . . General Operating Data

Long tons new feed per rougher spiral per hr	2.50
Long tons new feed per spiral per hr	1.43
Long tons new feed per day per sq ft floor space	2.47
(Includes only spirals which occupy 1163 sq ft of floor space for roughers and cleaners.)	
Size of feed, in.	minus ⅛
(8 pct minus 200 mesh)	
Wash water per spiral, gpm	5 to 7
Wash water total, gpm	420 to 588
Solids in feed, pct	30 to 45
Pulp feed, gpm	20 to 30
Circulating load, pct	10 to 20
Ratio of concentration, overall	1.89:1
Ratio of concentration, spirals	1.25:1
Total water used, gpm	2,600
Total water used per ton of feed (150 ton basis), gal	1,040
(Includes water lost in 78 in. Akins classifier overflow, ⅛ in. screen spray water, launder make-up water, pump water, and spiral wash water.)	

	Average Assays, Per Cent	
	Fe	SiO_2
Average 66 in. Akins classifier product (ball mill feed)	44.20	31.47
Spiral feed (deslimed abrasion mill product)	48.50	24.84
Spiral concentrate	55.05	14.83
Spiral tailing	36.39	43.01
Weight recovery (spirals only)	64.9	
Fe recovery (spirals only)	73.7	

coming to the spirals, this material is pumped to the 60 in. Akins classifier and loaded into the railroad cars without going to the spirals. The heavy-density concentrates then absorb the difference in grade to make a reasonably satisfactory final mill concentrate. The other extreme has occurred in which the plant crude consisted of too much minus ⅛ in. size material to treat in the spirals and the operator requests less feed. Fortunately these extremes are rare. If the tendency of the feed is to contain an abnormal amount of the upper size ranges, the pulp density is adjusted to about 45 pct solids in order to recover the coarse valuable particles. Conversely, if the feed tends to contain abnormal amounts of the finer sizes the pulp density is decreased to recover the finer sizes. By visual observation of the spiral tailing a good operator can readily determine adjustments needed.

Table 4 . . . Monthly Summary of Spiral Performance Assays, Per Cent

Product	May		June		July		August		September		October		Average	
	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2
Ball mill feed	41.47	35.12	46.36	28.21	43.41	32.16	45.67	29.67	42.68	33.89	42.40	34.75	44.20	31.47
Spiral feed	42.62	33.29	49.30	23.31	48.43	24.62	48.48	25.29	48.69	24.94	48.31	25.41	48.50	24.84
Spiral concentrate	51.45	19.71	54.98	14.67	54.55	15.35	53.94	17.05	55.60	13.04	56.10	13.26	55.05	14.83
Spiral tailing	37.88	40.42	41.11	35.65	35.11	44.73	34.45	46.33	33.50	47.79	33.65	47.07	36.39	43.01
Weight recovery, pct	34.94		59.04		68.87		71.98		65.76		65.30		64.90	
Iron recovery, pct	42.18		65.84		79.49		80.10		76.45		75.83		73.67	

That portion of the ball mill product overflowed from the 78 in. Akins classifier is not shown above but averaged about 25 pct Fe and 60 pct SiO_2. Note the progress made in the technique of operating the spiral installation. Excluding May, the spiral feed in general is consistent, while the spiral concentrate improves in grade without sacrificing additional iron in the tailing.

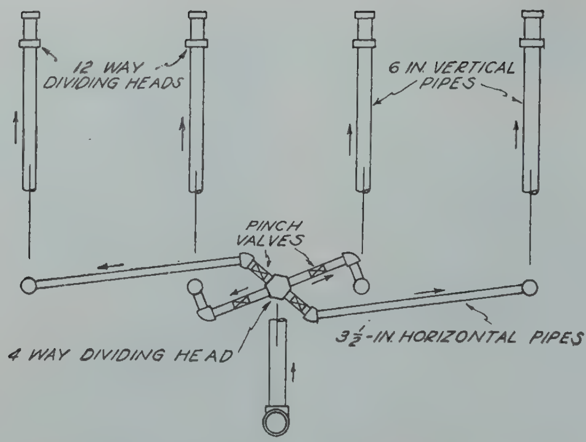

FIG 3—Feed distribution to four banks of first stage spirals.

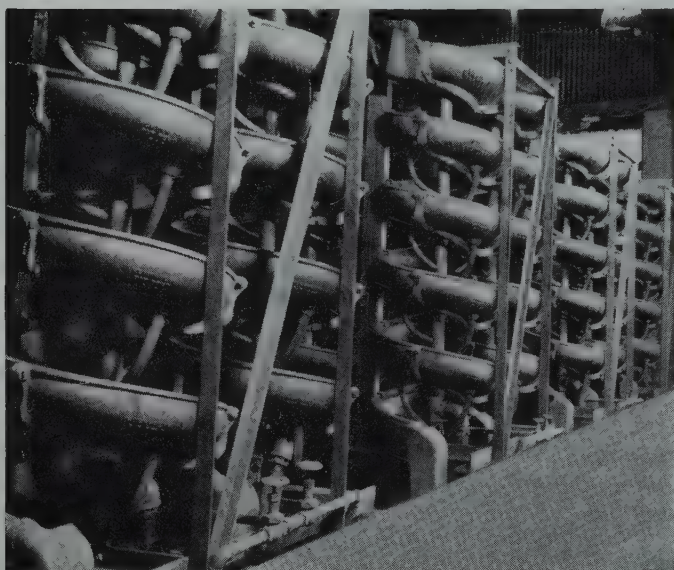

FIG 4—Part of the 48 first stage 5-turn spirals. Hill-Trumbull spiral plant of the Cleveland-Cliffs Iron Co.

Table 5 . . . Screen Analysis of Ball Mill Feed and Product

Size	Ball Mill Feed				Ball Mill Product			
	Wt, Pct	Cum Wt, Pct	Assays, Pct		Wt Pct	Cum Wt, Pct	Assays, Pct	
			Fe	SiO_2			Fe	SiO_2
+0.371 in.					0.52	0.52	56.8	10.8
−0.371 in. + 3 mesh	0.09	0.09			0.43	0.95	44.6	30.2
−3 + 6	2.36	2.45	52.3	18.4	1.76	2.71	53.4	15.5
−6 + 8	15.14	17.59	47.4	26.0	2.65	5.36	52.4	16.8
−8 + 14	23.66	41.25	46.5	27.8	8.96	14.32	50.2	22.1
−14 + 28	21.56	62.81	42.4	34.5	16.42	30.74	46.3	28.2
−28 + 35	11.13	73.94	39.9	38.7	10.30	41.04	42.3	34.1
−35 + 48	7.71	81.65	40.4	37.8	11.17	52.21	41.5	36.0
−48 + 65	6.15	87.80	41.5	36.4	9.00	61.21	41.8	36.1
−65 + 100	3.11	90.91	40.1	39.1	7.18	68.39	40.1	38.9
−100 + 150	2.41	93.32	32.1	51.5	4.63	73.02	35.2	45.9
−150 + 200	2.56	95.88	19.0	70.2	4.11	77.13	29.0	55.6
−200	4.12	100.00	17.4	72.7	22.87	100.00	35.1	54.3
Total	100.00	100.00	41.5	35.1	100.00	100.00	41.4	37.9

When the structure of the ore is such that the deslimed minus ⅛ in. ore reaching the spirals is of acceptable shipping grade it is sent directly to the concentrate thickener and the spirals are bypassed.

Data pertinent to spiral operation representative of the early part of the season of 1948 are given in Table 3. Average results by months are shown in Table 4.

GRINDING

The 6 by 10 ft Allis-Chalmers ball mill was fed 150 long tons of solids per hour. Preliminary laboratory tests on this ore showed a distinct advantage in both grade and recovery if the fine ore were given a differential grind. Grinding is fast and in open circuit. Table 5 shows the size distribution before and after abrasion grinding giving analysis of samples from a single shift and illustrating results obtainable from abrasion grinding of this ore. The table shows that at about 35 mesh the finer sizes begin to appear in larger weight percentages in the product than in the feed. Iron content also goes up in the finer sizes in the ball mill product indicating some grinding action on the locked iron and silica as well as on the silica alone. Silica content in the coarse sizes has been materially lowered as far down as 35 mesh. The bulk of the grinding work appears to be on the minus 6 plus 28 mesh sizes.

CLASSIFICATION

The 78 in. Akins classifier which receives the ball mill discharge helps concentrate the ore to a slight degree by desliming at about 200 mesh. Classifier overflow contains about 80 to 90 pct of minus 200 mesh solids and the product about 6 to 7 pct of minus 200 mesh solids. The overflow rarely contains more than 30 pct Fe.

SCREENING

Nature of the ore is such that the plus ⅛ in. particles after the abrasion grind are usually of sufficient grade to consider concentrate. The plus ⅛ in. particles come from spillage and screen cloth wear in other parts of the mill ahead of the spiral section. Screen oversize is usually just a few tons per hour and if an excess amount appears the spiral operator notifies the foreman so that the condition may be corrected. The screen cloth, 4 ft wide by 8 ft long and containing ⅛ by 2¾ in. slots, was tried with the long dimension across the line of pulp flow and parallel with the flow. When the slots were placed across the flow, screening was highly inefficient and it was nearly impossible to get good size separation at a rate of 120 tph. Turning the slots parallel to the flow immediately permitted easy and effective separation. A slotted screen is not the most desirable type to use for preparing a spiral feed. Flat ore particles are difficult to recover in the spiral as such particles when caught

by the fast water in the spiral tend to remain there and are discharged as tailing. A granular or blocky type of feed is more satisfactory unless, of course, the flat particles are gangue. Screen analysis of the ball mill product shows that the coarse sizes have a higher iron content than the fine sizes.

FEED DISTRIBUTION

To first stage spirals:

On leaving the first stage feed pump the pulp enters an 8 in. vertical pipe. A Humphreys dividing head with four horizontal 3½ in. pipe size outlets is located on top of the pipe. The outlets are branch lines, each containing the pulp feed for one bank of 12 spirals. These branch lines are horizontal 3½ in. pipes with pinch valves for closely equalizing and regulating the pulp flow to each bank of spirals. A 6 in. vertical pipe is attached to the end of each horizontal line and atop each 6 in. pipe is a 12-way Humphreys dividing head with 1¼ in. side outlets. Each 1¼ in. outlet provides feed for one spiral. Fig 3 and 4 illustrate the piping used in the distribution system of the first stage spirals.

To second stage spirals:

Pulp leaving the second stage pump enters a vertical 8 in. pipe. On top of the 8 in. pipe is a tee which is reduced to two 4 in. horizontal outlet pipes, splitting the feed evenly two ways. At the end of each horizontal outlet a vertical 6 in. pipe rises to an 18-way Humphreys dividing head with 1¼ in. side outlets. Each 1¼ in. outlet provides feed for one spiral. Fig 5 and 6 illustrate the piping system for feeding the 36 second stage spirals.

Spiral Concentration

Structure of the ore to be concentrated affects grade and recovery of spiral products. At the Hill-Trumbull plant two types of ore are treated in the spirals:

1. Hard dense hematite with very little limonite and at times containing some taconite. Very fine grinding would be required for complete liberation of silica from the iron oxide. The concentrate is characterized by a high silica to iron ratio. The plus 14 mesh sizes of this type ore contain many flat pieces of iron oxide while the silica is rounded or blocky.

2. Hard and soft hematite and limonite mixed with varying amounts of paint rock. Contained silica is liberated more easily from the iron oxide particles than in the other type ore. Spiral concentrate from this type ore usually contains less iron than the first type, but the silica content is lower. The reason for this is attributed to limonite in the concentrate.

It is well-known that the structure of the ore varies widely, but screen analyses are not available to illustrate the extent of this variation. Table 6 shows screen analyses of spiral products during one shift in the early part of the season. Fig 7 is a graph showing spiral performance during this period. Good recovery is shown in all sizes down to 200 mesh. Minus 200 mesh particles require a greater degree of cleaning than the coarse particles. Also the nature of spiral action is such that the extreme fines are difficult to recover. Good grade and recovery from these ores is obtained down to plus 28 mesh at which point the grade decreases and the recovery increases. Recovery begins to drop off in the minus 150 mesh sizes but the cleaning is fairly good.

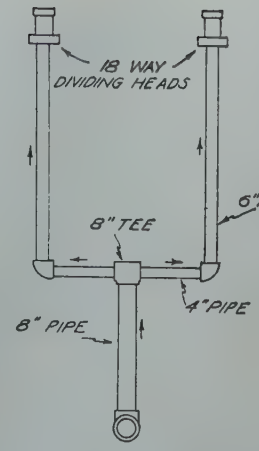

FIG 5—Feed distribution to two banks of second stage spirals.

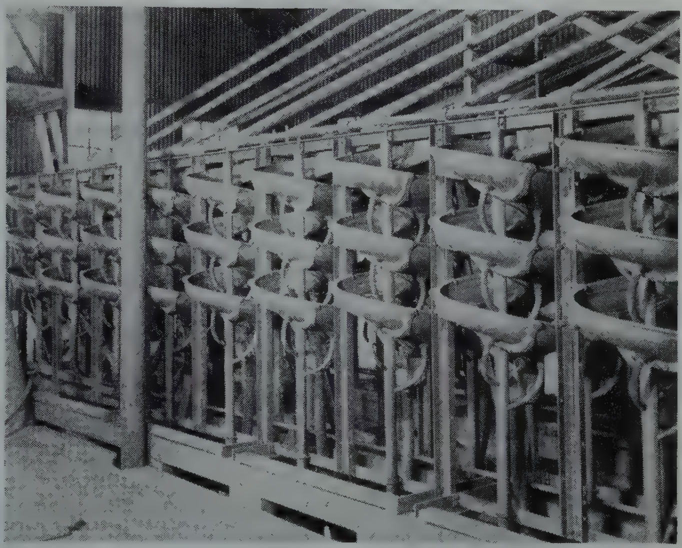

FIG 6—Part of the 36 second stage 3-turn spirals. Hill-Trumbull spiral plant of The Cleveland-Cliffs Iron Co.

In spiral concentration of an ore in which a large percentage of the weight and values is in the plus 35 mesh size, it is usually necessary to operate with a feed pulp density of about 40 pct solids and a pulp rate of 25 to 30 gpm. Both of these factors work against recovery of fine size particles. The high density is necessary to force the coarse iron oxide particles out of the stream and into the concentrate ports. The high pulp rate keeps the coarse gangue, fine gangue, and part of the fine iron suspended in the spiral stream.

The first stage concentrate indicates good grade would be obtained if all the plus 35 mesh sizes could be removed as a finished product. This can be done, in part, as a large portion of the coarser sizes are removed in the upper turns of the spiral. Removal of a finished concentrate in the first stage would be desirable for several reasons:

1. It is a good policy to remove a finished product as soon as possible. Second stage tailing contains some coarse iron recovered in the first stage. This tailing is returned to the first stage feed as part of a circulating load and hence is subject to possible loss in the tailing of the first stage operation.

2. The load of the second stage spirals would be lightened.

3. Circulating load would be decreased and the overall capacity of the spirals could be increased.

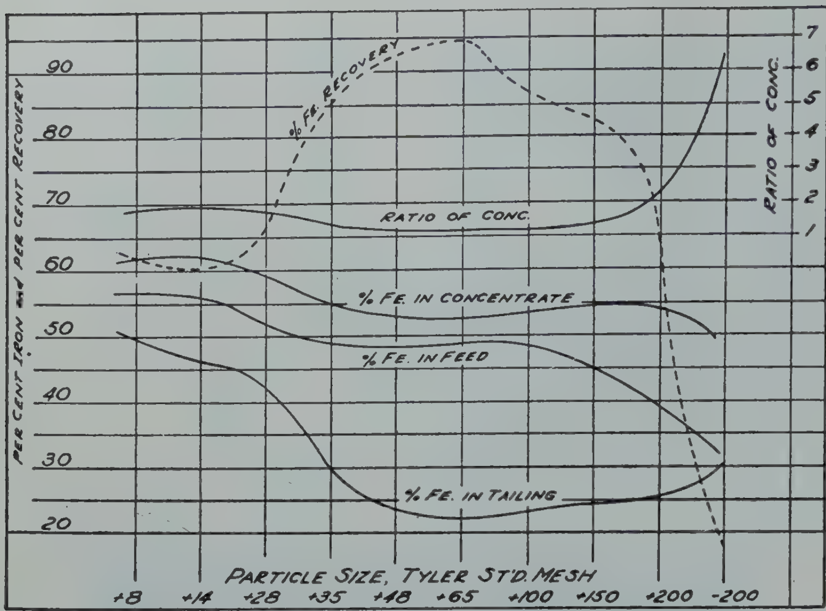

FIG 7—Distribution of particle size in spiral products.

4. Early removal of the coarse sizes would produce a closer sized product for the second stage feed. This would simplify spiral adjustments in the second stage and furnish a cleaner or higher grade product in the 28 to 65 mesh size range.

These items are under consideration as a possible flowsheet change.

It would appear from the screen analysis given in Table 6 that too wide a cut was made in the first stage on the intermediate sizes resulting in taking into the concentrate and middling too much silica and obtaining a low grade product. The zone of concentration is not sharp between the heavy and light minerals especially when the ore contains iron oxide particles with considerable range of specific gravity. Therefore it would seem better to keep the concentrate ports open rather wide to insure recovery of both coarse and intermediate sizes in the first stage. Limonite, paint rock, ferruginous silica, true middling, and porous iron particles having a specific gravity near 3.5 to 3.8 tend to intermingle with silica in the zone of concentration. If too narrow a cut is made the tailing is high in iron and the concentrate is not necessarily compensated or proportionately raised in grade. Removal of plus 28 mesh after the first stage of concentration would prepare a better suited feed for second stage concentration. Studies are being made to determine the best method for this removal.

An example of spiral results obtained when the grade of feed fluctuates widely between shifts is illustrated in Table 7.

Mill reports give no explanation for the high tailing obtained during the 11–7 shift although it would appear that the proper adjustments had been made on the 3–11 shift to compensate for the lower grade feed and then during the 11–7 shift when the grade of the feed increased, no adjustments were made. Such adjustments are necessary or more critical in ores of low concentration ratio, such as iron, manganese and chrome, than in other ores of higher concentration ratio because of the large weight percentage that needs to be removed from the ore

Table 6 . . . Screen Analyses of Spiral Products, Per Cent

Size	Feed (Screen Undersize)			True Spiral Feed			1st Stage Tailing			2nd Stage Concentrate			Recovery		
	Wt	Fe	SiO_2	Wt	Fe	SiO_2	Wt	Fe	SiO_2	Wt	Fe	SiO_2	Total Wt	Wt in Size	Fe in Size
+0.371 in.															
−0.371 in. + 3 mesh															
−3 + 6	0.95	60.20	9.20	0.19			0.18			0.74	62.80	6.30	0.56		
−6 + 8	3.68	58.30	12.10	2.85	58.90	10.90	5.00	48.40	25.50	2.87	62.90	5.80	2.19	68.3	73.69
−8 + 14	12.46	57.90	13.30	11.86	56.60	13.60	17.02	44.70	31.70	9.96	63.60	5.50	7.59	69.8	76.67
−14 + 28	21.90	52.10	21.40	20.98	51.50	21.40	22.30	44.50	44.50	18.85	59.00	12.30	14.38	52.4	59.34
−28 + 35	15.00	49.20	25.80	14.83	47.60	27.20	7.86	27.00	58.50	17.05	53.00	20.20	13.00	85.4	92.00
−35 + 48	15.03	50.10	25.05	13.22	47.50	27.70	3.98	21.10	67.40	18.14	52.20	21.80	13.83	93.5	97.42
−48 + 65	8.30	50.80	23.80	10.00	47.70	27.40	2.34	19.80	69.30	13.05	53.60	20.50	9.96	91.7	96.75
−65 + 100	8.10	51.50	23.25	7.44	48.00	27.70	2.96	22.30	65.90	10.02	55.10	18.30	7.64	89.0	95.22
−100 + 150	4.25	50.20	24.95	4.56	45.80	30.20	3.22	25.10	61.90	4.83	56.40	16.70	3.68	80.2	90.11
−150 + 200	3.56	44.80	33.10	3.33	40.80	37.70	4.96	24.10	63.20	2.82	56.40	16.90	2.15	64.1	80.70
−200	6.77	30.60	53.70	10.74	32.80	49.70	30.18	28.80	56.30	1.67	48.50	28.40	1.27	9.1	14.42
Total	100.00	50.44	24.24	100.00	47.96	26.90	100.00	33.01	49.61	100.00	55.87	16.74	76.25		84.45

Size	1st Stage Middling			2nd Stage Tailing			1st Stage Concentrate			60-in. Akins Overflow		
	Wt	Fe	SiO_2	Wt	Fe	SiO_2	Wt	Fe	SiO_2	Wt	Fe	SiO_2
+0.371 in.												
−0.371 in. + 3 mesh												
−3 + 6	0.07			0.36	50.70	22.00	0.75	62.20	6.70			
−6 + 8	1.07	55.70	15.20	1.82	50.80	22.10	2.74	62.00	7.00			
−8 + 14	8.75	58.30	11.10	14.75	54.00	17.60	11.83	62.00	7.20			
−14 + 28	22.06	53.30	18.80	33.63	47.30	27.70	21.92	55.60	16.30			
−28 + 35	16.57	43.50	33.30	16.13	38.70	41.10	16.41	50.30	24.00	0.05		
−35 + 48	16.12	37.40	43.00	10.74	34.10	48.20	16.47	50.10	24.45	0.30		
−48 + 65	12.10	35.30	46.50	5.99	38.20	49.50	11.97	51.70	22.75	1.89	31.30	52.40
−65 + 100	11.30	37.80	43.40	4.67	32.10	51.90	8.59	52.60	21.50	6.62	33.00	49.80
−100 + 150	3.83	41.50	37.90	2.46	32.20	57.70	3.84	53.00	21.30	10.57	37.30	43.70
−150 + 200	4.72	45.20	32.30	3.38	35.10	47.70	2.89	51.20	23.60	14.24	36.70	44.90
−200	3.41	42.30	36.70	6.06	31.20	53.60	2.59	40.20	39.65	66.33	27.60	58.15
Total	100.00	44.44	32.50	100.00	42.24	35.87	100.00	53.46	19.73	100.00	30.36	54.05

(Tables 8 and 9).

Example of shifts showing minimum and maximum recovery, not consecutive shifts, are given in Table 10.

Low recovery shown in Table 10 is probably due to early season inexperience in operation of the spiral plant and an overall average shows that as the season progressed the metallurgical results were increasingly better.

Summary

Use of the Humphreys spiral concentrator for treating minus ⅛ in. iron ore to produce an acceptable grade concentrate has been adopted and operated to the satisfaction of The Cleveland-Cliffs Iron Co. at the Hill-Trumbull plant on the Mesabi Range.

The spiral section of the plant, during the 1948 season, concentrated jig-type or retreat ores at an average rate of 120 long tons per hour and recovered about 78 tph of concentrate. Average grade of concentrate in October as the season ended was 56.10 pct Fe and 13.26 pct SiO_2.

Spiral metallurgy was satisfactory and it is believed that even better results can be obtained. Studies are being made for possible future changes.

The Cleveland-Cliffs Iron Co. has shown its desire to further the technology of iron ore beneficiation by its willingness to try new methods.

Acknowledgments

The authors are indebted to the entire organization of The Cleveland-Cliffs Iron Co. for its whole-hearted cooperation and especially to Messrs. W. A. Sterling, George Beasley, W. Van Slyke, Jack Chisholm, and William Johnson for their close cooperation and interest in making a success of the operation of this new method of iron ore beneficiation.

Table 7 . . . Spiral Results Obtained when Grade of Feed Fluctuates

Shift	Assays, Pct						Recovery, Pct	
	Feed		Concentrate		Tailing		Wt	Fe
	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2		
7–3	51.80	18.50	58.50	7.85	34.30	45.60	72.3	81.4
3–11	41.40	38.30	54.60	14.10	32.00	48.70	41.6	54.8
11–7	53.20	16.10	57.50	9.70	46.00	27.00	62.6	67.6

Table 8 . . . Example of Minimum Variation in Quality of Feed

Shift	Assays, Pct						Recovery, Pct	
	Feed		Concentrate		Tailing		Wt	Fe
	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2		
11–7	53.0	20.10	56.70	14.30	34.20	47.40	83.50	89.30
7–3	52.7	21.40	56.70	15.20	36.50	44.40	80.20	86.30

Table 9 . . . Example of Results Obtained from Low Grade Feed and High Grade Feed, Not Consecutive Shifts

Assays, Pct						Recovery, Pct	
Feed		Concentrate		Tailing		Wt	Fe
Fe	SiO_2	Fe	SiO_2	Fe	SiO_2		
35.7	42.30	51.10	17.65	23.00	62.30	45.2	64.7
56.7	16.00	59.50	12.50	41.80	36.60	84.2	88.4

Table 10 . . . Example of Shifts Showing Minimum and Maximum Recovery

	Assays, Pct						Recovery, Pct	
	Feed		Concentrate		Tailing		Wt	Fe
	Fe	SiO_2	Fe	SiO_2	Fe	SiO_2		
Low recovery....	52.30	17.90	56.50	11.60	50.50	20.70	30.0	32.4
High recovery...	52.30	15.50	54.50	12.70	37.70	39.65	88.1	91.4

Formation and Properties of Single Crystals of Synthetic Rutile

By CHARLES H. MOORE, JR.,* Member AIME

Introduction

In the study of the properties of rutile pigments it became apparent several years ago that certain physical and optical properties could not be determined on particles of pigmentary size. Since reflected light is the dominant type which reaches the eye from small particles, the true color of pure rutile was not known. Most rutile pigments are acicular in habit, elongated parallel to the "*c*" axis. It was considered important to know the nature and tone of light transmitted, for example, by a basal plate. Further, as shown by spectrophotometric curves, there is a very strong absorption of light of 4000 Å as measured on particles with random orientation. It would be interesting to see whether this absorption position varied with predetermined and selected orientations. Also, since the behavior of titanium pigments in a vehicle is important, a knowledge of the surface wettability of single crystals by various media should prove very fruitful. Finally, the optical properties of rutile are such that it should show, in a single pure crystal, greater fire and brilliance than the diamond. Mineralogists had long speculated that this would be the case and curiosity was strong to produce a single crystal large enough to cut.

In light of the above, a project was initiated for the formation of single crystals of rutile.

Method of Formation of Rutile Single Crystals

Fortunately, purification of TiO_2 produced from both titanium tetrachloride and titanyl sulphate solutions had reached the stage where spectrographically pure starting material could be made. Spectrographic analysis of the feed material used in the initial experiments is compared with the present feed material in Table 1. It is apparent that the present large scale purification is approximately as good as the original experimental, small scale purification.

Verneuil[1] produced synthetic ruby and corundum in 1904 by the well-known flame fusion process, which today bears his name. Except for mechanical improvements the present commercial production of synthetics by flame fusion is essentially unchanged from his initial method. His burner consisted of an inner orifice through which was fed the feed material and oxygen. Surrounding this was a larger outer orifice through which was fed hydrogen at lower pressure. This arrangement consists then of a lance of oxygen burning in an atmosphere of hydrogen. The proportions of hydrogen to oxygen usually used in corundum production is about 3:1, providing there is sufficient temperature to reach the 2050°C required for the fusion of alumina.

From work done in the Titanium Division Laboratories[2] and from the published work of Erlich[3] it was known that rutile appears to lose oxygen near its melting point and that it would not be possible to grow single crystals in a strongly reducing environment. Consequently, in order to determine whether single crystals could be grown at all, an Airco oxygen-acetylene torch with

Table 1 . . . Comparison of Original and Present Feed Material

	Original Feed Material	Present Feed Material
SiO_2	<0.02	0.04
Fe_2O_3	<0.001	<0.001
Al_2O_3	<0.001	0.001
Sb_2O_3	<0.002	<0.002
SnO_2	<0.001	<0.001
Mg	0.0005	0.0005
Cb	<0.005	<0.005
Cu	<0.0001	0.0001
Pb	<0.002	<0.002
Mn	<0.00005	<0.00005
W	<0.005	<0.01
V	<0.0005	<0.002
Ca	<0.01	
Cr	<0.0001	
Ba	0.001	

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[1] References are at the end of the paper.

twin nozzles was set up as shown in Fig 1. A hopper, consisting of a 6 in. length of mullite furnace muffle 1½ in. in diameter was mounted vertically, with the bottom covered by a 100 mesh wire screen. A glass funnel leading to a copper tube was mounted below the hopper, with the end of the feed tube terminated an inch above the junction of the flames from the two tips. The feed material was vibrated through the screen by means of a small magnetic vibrator attached to the side of the hopper.

The tips of the torch were bent 45°, so that they faced each other. They were wound with $\frac{1}{16}$ in. copper cooling coils and inserted through holes cut into a porous refractory brick. A hole was cut in the center to admit the feed material and the brick cemented to the torch with sodium silicate cement.

A preliminary furnace shell with an inside diameter of 2 in. was cast from the standard clay mix and the brick top with the torch inserted was rested upon it. In order to lower the fused material away from the flame, the furnace assembly was placed upon a scissors automobile mechanical jack, and the furnace lowered by hand. Small crystals, approximately 1 to 2 mm in diameter and 4 to 5 mm in length were grown with this apparatus. This showed that in spite of its tendency for very rapid crystallization rutile single crystals could be grown by the flame fusion method. It was obvious, however, that with a point fusion zone, no crystals of appreciable size could be grown and that any crystals produced would be so badly strained as to be useless. In consequence we designed a burner which would provide the minimum reducing environment and which would enlarge the area of constant temperature, to yield a broader fusion zone without a horizontal thermal gradient.

Where hydrogen and oxygen burn, an intense heat is created at the zone of mixing of the two gases. Turbulence of the gases broadens the zone but it is still a single reaction area. However, if an envelope of oxygen is made to surround the hydrogen to yield a three component, or three envelope, nozzle burning flame, there are two reaction zones with sufficient turbulence so that a constant temperature is maintained over the effective diameter of the nozzle. In addition, the outer oxygen envelope prevents excessive reduction of the TiO_2. A 0.5 in. nozzle diameter burner was built according to this design and single unfractured crystals of pure rutile, 1 in. long and 0.45 in. in diameter, were grown. The burner is virtually fool-proof, and the only critical factors are the relative space velocities of the three gas streams.

In order to prevent periodic cooling of the flame by the showers of feed material produced by the customary tapping of the hopper, a very sensitive hopper was designed and a vibrator employed in place of the tapper commonly used on commercial burners. The constant flow of feed not only prevented the strain in the crystal which had formerly resulted from periodic cooling of the flame, but also eliminated the layer structure characteristic of Verneuil Process synthetic

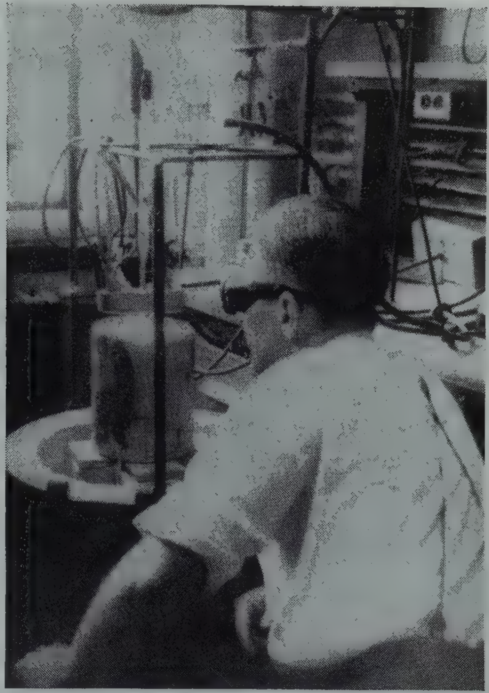

FIG 1—The first apparatus for attempted growth of single rutile crystals.

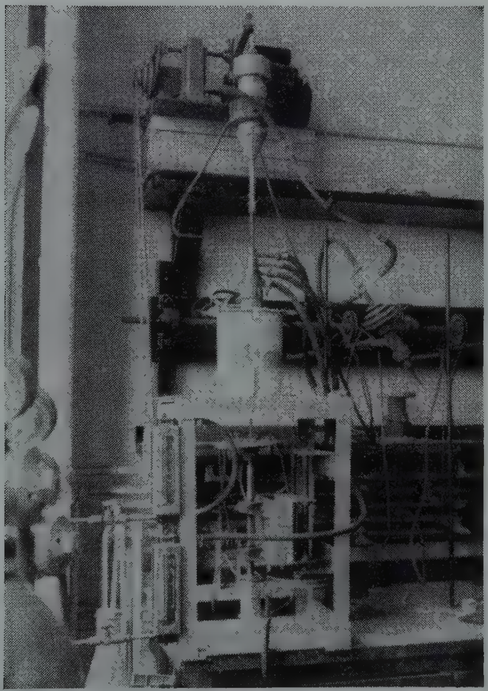

FIG 2—First burner made according to Titanium Laboratory design for rutile crystal growth.

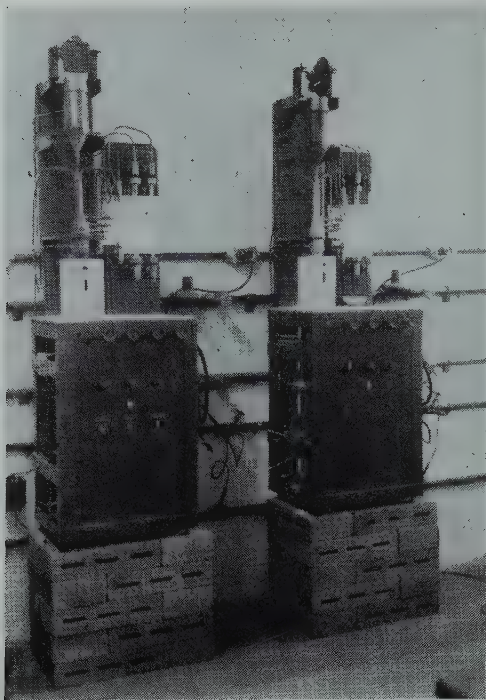

FIG 3—Improved burners and apparatus currently in operation for rutile single crystal growth.

FIG 4—Rutile single crystals showing improvement in single crystals. Early growth is illustrated by the two boules at left and current production by the three boules at right.

crystals. This layer fusion had been deemed by Verneuil as an essential part of the flame fusion technique.

It was also felt that the hand lowering of the growing boule must contribute to strain in the crystal, since any operator must of necessity lower the boule a minimum of several millimeters if he is to keep up with the rate of boule growth and hold the fusion zone at a constant position in the flame. Therefore, a variable speed motor was attached to a worm gear and the pedestal lowered automatically. A provision was made also for a variable speed rotation of the pedestal in the event this was found to facilitate crystal growth. This original design burner and apparatus is illustrated in Fig 2.

The control of flow of oxygen and hydrogen to the burner is of the greatest importance. This was accomplished, in the 0.5 in. burner, by reduction valves on the tanks followed by low pressure diaphragm valves in the line; and which were followed in turn by needle valves in front of the rotometers. A simultaneous shut-off valve also was used. A filter of Spanish moss placed in the line removed excess moisture from the gases but is not considered essential when gases of normal purity are used. Pigtails connecting two tanks each of hydrogen and oxygen gave sufficient capacity of gas so that a 6 hr run could be made without shutting off, and without marked gas fluctuation.

The final burner design is essentially the same, except for engineering improvements, as the 0.5 in. nozzle burner. The present burners are 0.75 in. nozzle diameter and on which the hopper and lowering mechanism have been materially improved. These burners are connected to fixed lines which are fed from a series of 5 hydrogen and 5 oxygen tanks (see Fig 3).

Mechanism of Growth

A congruently melting compound which has a melting point within the range of the temperature of combustion of hydrogen and oxygen is quite simple to grow as a single crystal. The only absolutely essential features are accurate control of gases, a burner which will yield a fusion zone with a low horizontal thermal gradient, and a perfectly straight spindle under the pedestal. The control of particle size and shape is essential for a commercial operation, since the particles must be absorbed into the molten surface and melted completely as rapidly as possible.

Single crystals of rutile are grown in the following manner. The burner is cut on, the sintered alumina furnace (2½ in. id) is closed and the chamber allowed to heat for several minutes. The pedestal, consisting of any sufficiently refractory ceramic material, usually alumina or stabilized zirconia, is raised until it is just below the hot zone of the flame, which has a very steep vertical gradient. The vibrator is then turned on and the feed material drops on the pedestal. The temperature of the flame is held below the melting point of the compound by holding the hydrogen content of the flame low. A steep sintered cone is built up until the tip is approximately 1 mm across. The hydrogen flow rate is then increased and the tip melted. If frozen at this point the tip is found to consist of 3 to 5 crystals with various orientations. If not frozen, it is allowed to grow up into the hottest zone of the flame. It is essential at this stage that the flow of feed be very concentrated, since the "foot" must be kept narrow. This is necessary in order for one crystal to assume dominance and "freeze out" the others. After the foot has grown to a height of 4 to 6 mm, which takes about 5 min, the gas velocity is increased, the automatic lowering turned on and the width of the growing crystal is increased. This increase in velocity is continued until the desired width is obtained. The lowering rate is then synchronized with the rate

of crystal growth and the mechanism is allowed to run until a boule of requisite size is attained, when all gases are cut off simultaneously. After cooling in the furnace for 30 min the boule is removed. Although boules of large size can be grown, because of the increase in strain with size, the boules are restricted to approximately 100 carats (see Fig 4).

Growth from a seed crystal is much simpler since the very critical "foot" stage is eliminated. However, this is not considered a desirable technique for commercial production because of the manpower-time requirements. Seed are usually cemented to the pedestal by an alumina-clay cement. The crystallographic orientation obtainable by growth from a seed does not appear necessary with rutile. Crystals grown from cones have the "*c*" axis less than 20° off the vertical axis of the boule.

Characteristics of Single Rutile Crystals

Rutile single crystals grown by the above process are opaque black when removed from the furnace. However, because of the outer oxygen envelope, the rutile structure holds and there is no increment of Ti_2O_3 in the entire structure. A measurement of the unit cell of this black material is nearly identical with that of the fully oxidized rutile structure, both having a tetragonal unit cell. The colorless crystals show 4.5815 and 2.9509 Å, and the black shows 4.5820 and 2.9510 for the *a* and *c* directions respectively.* This black material is a strong semiconductor, and has a specific gravity of 4.268. It is possible to convert this material to colorless rutile by heating in a stream of oxygen. The oxygen addition to the crystal is directly proportional to the size of the crystal being reoxidized and the temperature used. The degree of reoxidation is readily checked by color changes, as the material goes from black through deep blue to light blue to green to colorless with a yellow tone. The specific gravity of the clear material has been found to vary between 4.29 and 4.39.† As will be shown, this variation in optically identical material was also noted in the dielectric measurements.

Table 2 shows hardnesses of the oxidation states of TiO_2 single crystals, measured by the diamond indentor, commonly known as the Knoop microindentor.

The synthetic material as measured by the Knoop microindentation method is universally harder than natural rutile, in that the latter is reported having a hardness of 6 to 6½ Mohs scale, while the synthetic is as hard or harder than quartz in all directions. However, hardness measured in the direction of the "*c*" axis on the prism face exhibited a "butterfly" fracture effect around the indentation. This indicates a relief of stress in a sharp angle cone, with the length of the indentation correspondingly shortened. This is brought about by an excessive brittleness in this direction, making the indentation measurements in the direction of the "*c*" axis unreliable. Such a phenomenon would be expected to occur if there exists an atom deficiency in the planes parallel to "*c*" leaving many broken bonds, or if the atoms are slightly displaced along these planes, creating a state of strain.

Table 3 shows the dielectric measurements of single rutile crystals. These data show a marked change in dielectric properties, corresponding to the degree of "reduction" of the crystals. The measurements were made by the Crystal Section of the Naval Research Laboratory,[4] using a General Radio 716 capacity bridge in conjunction with a simple sample holder. All samples were colorless material reduced by H_2 at 600°C for the various time intervals.

Table 2 . . . Hardness of TiO_2 Crystals by Knoop Microindentation Method

Sample	Orientation: Basal Plane Perpendicular "c" Axis — Parallel One Prism	Basal Plane Perpendicular "c" Axis — Parallel Other Prism	Prism Face Parallel "c" Axis — In Direction of "c" Axis	Prism Face Parallel "c" Axis — Perpendicular to "c" Axis	Pyramid Face Cut, 45° to "c" Axis	Standard: Mineral	Mohs Hardness	Indentation No.
Perfectly orientated colorless cube	898	898	805[a]	840		corundum	9	±1,800
Yellowish over-oxidized crystal					940	topaz	8	±1,200
Light blue partially reduced cube	750	850	831[a]	835	890	quartz	7	790 710
Dark blue cube	766	840	859[a]	885		orthoclase	6	600–690
Black	940	950	1,000[a]	767				

[a] Indicates erroneous readings caused by excessive brittleness.

Table 3 . . . Change of Properties Upon Reduction

No. of Reduction Exposure	Crystallographic Direction	Oriented Cube: Dielectric Constant	Oriented Cube: Resistivity (ohm cm)	Rectangular with Long Direction Parallel to "c" Axis: Dielectric Constant	Rectangular with Long Direction Parallel to "c" Axis: Resistivity (ohm cm)	Rectangular with Long Direction Parallel to "c" Axis: Dielectric Constant	Rectangular with Long Direction Parallel to "c" Axis: Resistivity (ohm cm)
0	a a c	655 275 83.5	3.14×10^9 4.5×10^{10} 1.55×10^{13}	149	15.0×10^{12}	187	3.79×10^{12}
1.	a a c	4.9×10^4 3.88×10^4 3.16×10^4	1.7×10^5 4.82×10^5 6.33×10^3	4.68×10^4	1.13×10^5	2.34×10^4	4.73×10^5
2.	a a	5.7×10^4	3.82×10^4 1.92×10^4 3.0×10^4	5.35×10^4	2.35×15^6	3.01×10^4	7.12×10^5
3.	a a c	4.58×10^4 4.58×10^4	4.2×10^4 1.55×10^4	4.15×10^4	3.39×10^5	2.22×10^4	9.5×10^5
4.	a a c	4.42×10^4 4.24×10^4 3.64×10^4	1.275×10^5 2.24×10^5 1.55×10^5	2.95×10^4	2.82×10^6	1.59×10^4	2.37×10^7
5.	a a c	4.31×10^4 4.37×10^4 4.8×10^4	2.55×10^4 4.82×10^4 2.64×10^4	3.17×10^4	2.62×10^6	1.62×10^4	1.66×10^6
6.	a a c	4.2×10^4 3.53×10^4 4.7×10^4	2.12×10^4 7.06×10^4 2.18×10^4	3.22×10^4	2.82×10^5	1.79×10^4	9.47×10^5
7.	a a c	4.25×10^4 3.88×10^4 5.67×10^4	 8.02×10^4 4.97×10^4	3.27×10^4	9.42×10^5	1.81×10^4	9.47×10^5
8.	All samples indicate strong decay of dielectric constant during measurement.						
9.	Cannot measure dielectric constant—resistivity too low.						

* Measurement made by W. F. Sullivan of Titanium Division, Research Laboratory.

† Measurements were made by the volume displacement method.

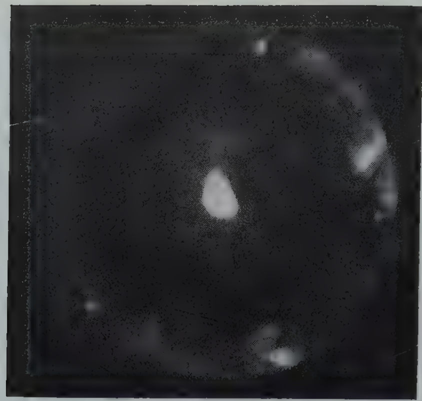

FIG 5—Stone cut with table face perpendicular to "c" axis. Culet appears without effect of birefringence. (× 10 magn.)

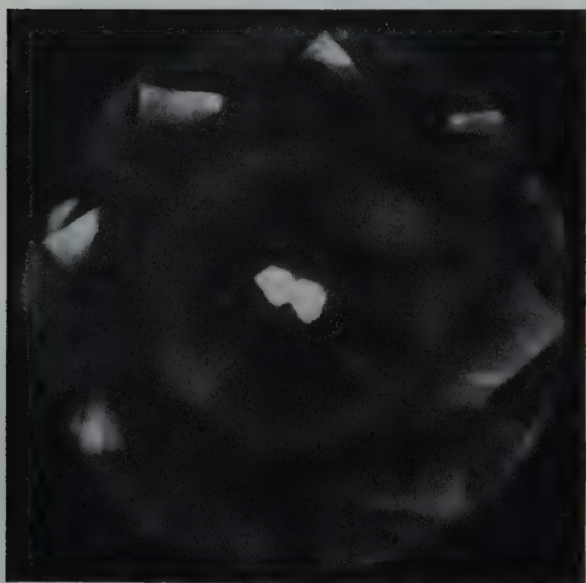

FIG 6—Stone cut with table face at 45° to "c" axis. Birefringence causes culet to appear at two overlapping openings. (× 12½ magn.)

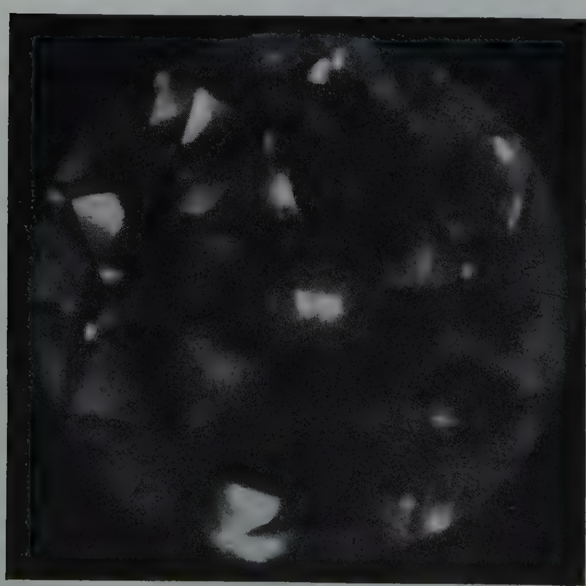

FIG 7—Stone cut with table face parallel to "c" axis. Birefringence now causes culet to appear as two distinct openings. (× 12½ magn.)

The samples were held at 5 min intervals for the first five steps, 10 min intervals for the next two steps and 30 min intervals for the last three steps (with the latter held at 700 to 800°C). It was observed on making the reduction tests on the oriented cube, that the cube was colored a faint blue in the region of the basal faces after 10 min while still colorless in the central portions. As reduction time increased the blue zone moved into the crystal from both basal planes. This supplements the evidence shown by the hardness tests that the greatest loss of the oxygen upon reduction is in the direction of the "c" axis. Since TiO_2 is tetragonal, one would expect, if the change from colorless to black is caused by loss of planes of oxygen atoms, that this loss would be in a preferred direction. This is indicated by the evidence.

In light of the physical data, certain facts are noted concerning the single crystal rutile structure:

1. It is possible to vary these crystals from clear to opaque black without apparently affecting the volume or dimensions of the tetragonal unit cell.

2. There is a distinct difference in bonding between the planes of atoms parallel to the "c" axis and those in other directions.

3. Differences in hardness, brittleness, specific gravity and dielectric constant occur in crystals which are identical in color.

4. A given crystal can be taken through a complete cycle from colorless to a black semiconductor, apparently identical with the original furnace product, and back again to colorless, at least four times without materially affecting its properties. After six to eight cycles, the crystal becomes embrittled. The ability of the crystals to reduce and oxidize without structural change indicate an elasticity of structure not generally recognized.

The concept that removal of oxygen atoms brings about a darkening of the color upon reduction, to yield an increasing increment of Ti^{+3} may be correct, but it is difficult to apply to oxidation. A structure for the clear rutile analogous to FeO and FeS_2 should be considered as a possibility. These compounds exist as stable lattices with nonstoichiometric proportions of constituents. The strong covalent bonds necessary for such a cation deficient structure to exist have been shown, by dielectric studies, to occur in rutile.[5]

Rutile pigments show a strong ab-

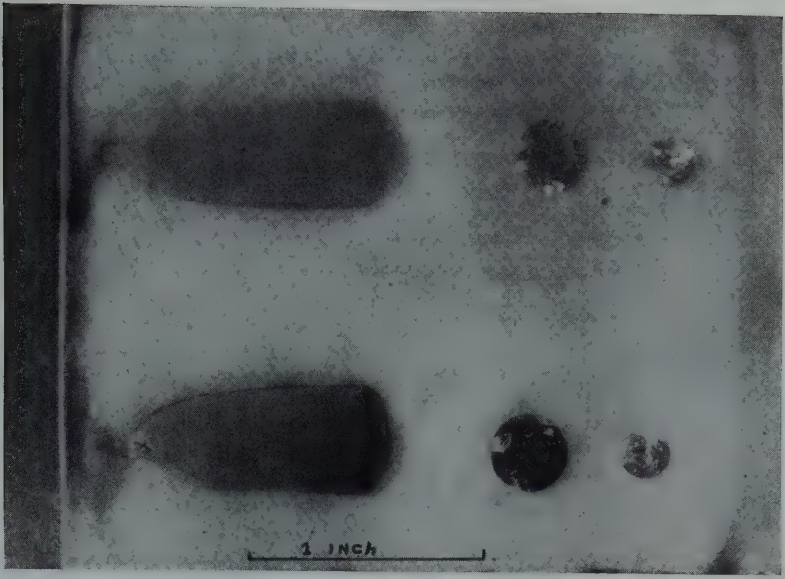

FIG 8—Two half boules, the lower one dark blue and the upper one completely oxidized, with stones cut from each type. Smallest stones are 1.5 carats.

sorption of light at 4000Å to yield a higher increment of longer wavelengths in the transmitted light which results in a yellow tone. Single crystals of rutile are opaque to all wavelengths of light between 2100 and 3200Å and transparent from 4100 to 6800Å at 77°K.[6]

Synthetic Rutile As Gem Stones

The indices of refraction of synthetic rutile, measured by a three circle goniometer on a 20° prism cut parallel to the "*c*" axis showed 2.605 ± 0.004 for the ordinary ray and 2.901 ± 0.004 for the extraordinary ray. Sodium light was used. Measurements made by B. W. Anderson and C. J. Payne[7] on a cut rutile gem show very similar results. These values are within the range of the refractive indices of natural rutile. The refractive index, the marked dispersion, and the extremely high birefringence yields a brilliance and fire unrivalled by any other gem. The birefringe gives rise to a marked phenomena in the cut stones as illustrated by Fig 5 to 7. These photographs are taken of cut and faceted stones cut perpendicular to "*c*" axis, 45° to "*c*" axis and parallel to "*c*" axis, respectively. The well-known illustration of the double refraction for different optical orientations is clearly shown by the single, overlapping double and apparently separate culets at the base of the stones.

Even though the stone measured by Anderson and Payne was not cut to yield maximum values, the dispersion of the lowest index of refraction (0.2851) is six times that of a diamond. This dispersion of the extraordinary ray is even higher. This, of course, accounts for the tremendous "fire" of the material.

The standard brilliant or diamond cut is satisfactory for rutile. Since this type of cutting is designed to give total reflection of light by the facets of material with a critical angle of 24° (diamond) it will, of course, also serve for material with a critical angle of a maximum of 18° (rutile). However, if desired, the 6° advantage rutile has over diamond could be utilized to allow a wider bottom internal angle (100° is critical for diamond) yielding a larger diameter stone for its weight. Fig 8 shows boules and cut stones of synthetic rutile.

Since the colorless rutile is not a duplication or imitation of a natural stone as are the other synthetics, it is probably the first truly new gem since the advent of modern jewelry. It is the first material whose optical properties are such that it is superior to diamond in both brilliance and fire. Its future in this industry at least seems assured.

Acknowledgment

The writer wishes to express his appreciation to Dr. R. Dahlstrom, Technical Director, Titanium Division, National Lead Company, who made this investigation possible; to Drs. Paul Egli and S. Zerfoss and the staff of the Crystal Section of the Naval Research Laboratory for the electrical measurements; and to H. F. McMurdie of the National Bureau of Standards for refractive indices determinations. He also wishes to thank Dr. B. Levin, Chief of the Microoptical Section, Squier Signal Corps Laboratory, Fort Monmouth, N. J., for use of their equipment for the hardness testing.

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Coal Washing in Washington, Oregon, and Alaska*

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Coal washing assumed an important role in the mining industry of the Pacific Northwest long before washing practice became firmly established in the Appalachian field. A Scaife washer was operated in the state of Washington in 1887; and by 1927, the first year for which complete statistics were compiled, nearly a third of the state's coal production was washed, in comparison with only about 5 pct for the country as a whole. Mechanical preparation was adopted in Alaska in 1922 in a plant constructed by the United States Navy at Chickaloon to provide bunker coal for naval forces operating in northern waters.

Many of the coal beds mined in Washington and Alaska contained more impurities than those mined elsewhere, and this circumstance contributed to the early interest in mechanical cleaning. A much more important factor, however, was the inclination of the coal beds. With steeply pitching beds, hand sorting by the miner at the face is impossible; consequently, all material must be loaded and dealt with on the surface, just as is now proving the case with mechanical mining of flat beds. Thus "full-seam" mining afforded the same stimulus to the early development of coal washing in the Northwest that it is providing under mechanization in the rest of the country today.

Object and Scope

This report is intended to provide a general summary of the status of coal-washing practice in Washington, Oregon, and Alaska—an area in which some of the coals mined are more difficult to wash than those mined elsewhere in the country. Detailed flowsheets of plants, statistical data on production, and cost figures have been omitted, largely for the sake of brevity but also because such information is principally of only local interest. On the other hand, considerable washability data have been included because they are required to indicate the type of washing problems encountered in this area.

San Francisco Meeting, February 1949.

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[1] References are at the end of the paper.

Washington

At present Washington leads all other states in the percentage of its total production that is cleaned mechanically, 82.1 pct in comparison with the national average of 25.6 pct in 1945.[1] The washing problems encountered in this state are highly variable because the mountain-building forces that created the Cascade Range caused such intense folding and faulting of the beds in some fields that the rank of the coal was increased to anthracitic, while in other fields more distant from the mountains the beds were relatively undisturbed, and the rank of the coal ranges down through subbituminous to lignitic.

The complicated and burdensome schedule of sizes prepared in most coal fields has never been required by the Washington market. With but few exceptions, lump is prepared on a round-hole screen of about 3 in. size, egg is about 3 to 1⅝ or 1½ in., and nut is 1⅝ or 1½ in. to either 1 or ¾ in. Some mines screen no finer than ¾ in. and market a slack coal of that top size, but generally the coal is screened at about ¼ in. to give a stoker coal of ¾ or 1 to ¼ in., and a ¼ in. to 0 "buckwheat." During recent years the market for lump coal has largely disappeared, and lump is crushed to supplement the pro-

duction of stoker coal at most mines. In fact, at many operations shipments of lump coal rarely exceed 5 pct of the output.

ROSLYN-CLE ELUM FIELD

The areas of the Roslyn-Cle Elum field worked up until recent years are characterized by beds dipping only 10 to 15°, and the principal bed, the Roslyn, is relatively clean. Thus mechanical cleaning was not adopted in this field until 1935, when mining on steeper pitches and the introduction of mechanical loading in the flatter areas had progressed to a point where the amount of impurity hoisted with the coal was excessive. In that year cleaning plants were built by the two principal companies operating in the field, the Northwestern Improvement Co., coal-producing subsidiary of the Northern Pacific Railway, and the Roslyn-Cascade Coal Co.

Coal from the Roslyn bed is high-volatile A bituminous in rank. Considerable shale, clay, and a moderate amount of bony material contaminate the run-of-mine product, but it contains less than 1 pct of sulphur. Roslyn-bed coal, unlike many of the coals from western Washington fields, is only moderately difficult to wash. The quantity of intermediate-density material, on which the difficulty of washing chiefly depends, is, for example, less than that in the Thick Freeport bed of Pennsylvania but more than that in the No. 6 bed of southern Illinois. A specific-gravity analysis of 3 in. to 0 raw coal from the Roslyn bed is shown in Table 1.

Table 1 . . . Specific-gravity Analysis of Raw 3 In. Slack from Roslyn Bed

Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Cumulative	
			Weight, Per Cent	Ash,[a] Per Cent
Under 1.30	46.5	6.6	46.5	6.6
1.30 to 1.40	38.1	12.4	84.6	9.2
1.40 to 1.60	7.2	27.1	91.8	10.6
1.60 to 1.80	1.8	47.4	93.6	11.3
Over 1.80	6.4	75.7	100.0	15.4

[a] Moisture-free basis.

The central cleaning plant of the Northwestern Improvement Co. was built to serve three mines, Roslyn Nos. 3, 5, and 9; it has a capacity of 200 tons per hour. At the individual mine tipples, the raw coal is screened at 3 in. round-hole size on shakers, the oversize hand-picked, and the clean lump crushed and added to the 3 in. screenings to constitute the washery feed. At the cleaning plant the raw coal is prescreened on a vibrator to give 3 to 1⅝ in. egg, 1⅝ to ¾ in. nut, and ¾ in. to 0 slack. Each of these sizes is washed separately in Vissac, pulsator-type jigs of 60 tph capacity; separate single jigs serve for the egg and nut sizes, and the ¾ in. slack is split between two units. The washed product from two slack jigs is screened on a vibrator dressed with ¼ in. cloth and the undersize rewashed on a group of six Deister-Overstrom tables.

A refuse retreatment circuit was installed in the plant several years ago. Refuse from the egg and nut jigs is crushed to 1 in. top size and, together with the refuse from the slack jigs, is rewashed in a small Elmore jig. Washed coal from the Elmore jig is crushed to pass ¼ in. and joins the ¼ in. to 0 material screened from the washed slack to form the feed for the wet tables. Final washed coal from the tables is recovered in a drag-type settling tank and dewatered in a centrifugal drier; the effluent of the centrifugal drier is returned to the fine-coal circuit by way of the usual conical settling tank. The slack coal is further dewatered in a Vissac heat drier during the winter months.

After washing, the egg and nut sizes are rescreened on shakers to remove degradation material. Egg, nut, 1⅝ in. slack, and ¾ in. slack are the sizes regularly shipped, and much of the coal goes for railway use. The plant is operated under laboratory control, with samples for ash analysis collected from each car of coal shipped.

The second washing plant in this field, operated by the Roslyn-Cascade Coal Co., treats coal from the No. 6 bed. As shown by the specific-gravity analysis in Table 2, this coal is somewhat more bony than that from the Roslyn bed.

Table 2 . . . Specific-gravity Analysis of Raw 3 In. Slack from No. 6 Bed

Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Cumulative	
			Weight, Per Cent	Ash,[a] Per Cent
Under 1.35	60.3	9.5	60.3	9.5
1.35 to 1.50	23.4	20.7	83.7	12.6
1.50 to 1.70	5.7	37.4	89.4	14.2
Over 1.70	10.6	74.7	100.0	20.6

[a] Moisture-free basis.

The run-of-mine product is screened at 3 in., the oversize lump hand-picked, and most of the clean lump crushed in a pick-breaker to join the 3 in. screenings constituting the washery feed. A 5-cell, 2-compartment, Baum-type jig of 80 tph capacity is used for cleaning 3 to ¼ in. coal, and a Stump air-flow unit is used for the ¼ in. to 0 slack. After washing, the coal is sized on shakers to produce 3 to 1⅝ in. egg, 1⅝ to ¾ in. nut, and ¾ in. slack, the sizes normally shipped. A Ruggles-Cole heat drier is used for reducing moisture in the ¾ in. slack during the winter months, and this product can be oil-treated for dustproofing if desired.

WESTERN WASHINGTON FIELDS

About 30 mines, most of them small truck operations, are active in western Washington at present, and over two-thirds of this number have washeries. The flowsheets employed at the small mines are almost identical; lump is prepared at 3 in. on a shaker, and the 3 in. slack is washed in a Forrester jig. In fact, the Forrester jig is standard equipment at small mines in the state. A single-cell, piston-type unit, the Forrester relies on manual control of refuse removal and hence is "only as good as the operator."

Largest of the western Washington plants, the washery of the Bellingham Coal Mines, located at Bellingham, has a capacity of about 1000 tons per day, using Forrester jigs for washing 4 to 1⅛ in. coal, a Faust jig for nut coal, and wet tables for material finer than 5⁄16 in.

A 5-cell, 2-compartment, Baum-type jig installed several years ago at the Black Diamond washery of the Palmer Coking Coal Co. is employed for the 3 in. slack from several small mines operated by the company. Coal from a strip pit on the No. 12 or Fulton bed, a specific-gravity analysis of which is shown in Table 3, is particularly difficult to wash. This difficulty is illustrated in Fig 1 which compares the Fulton coal with the Roslyn and No. 6 bed coals previously described. It contains nearly 25 pct of impurity heavier than 1.70 sp gr and a large proportion of intermediate-density material. Moreover, much of the impurity is clay of such a plastic character that small particles adhere tenaciously to pieces of clean coal and can be removed only by vigorous sprays on the washed-coal sizing screens.

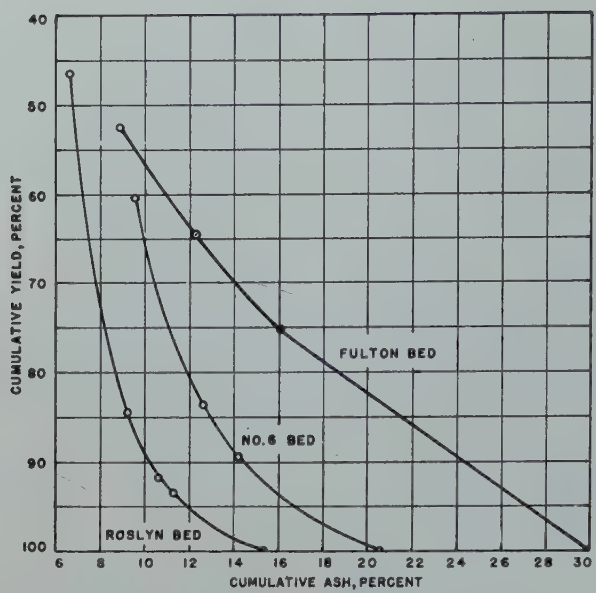

FIG 1—Yield-ash curves for Roslyn, No. 6, and Fulton beds.

Table 3 . . . Specific-gravity Analysis of Raw Slack from Fulton Bed

Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Cumulative	
			Weight, Per Cent	Ash,[a] Per Cent
Under 1.40	52.4	8.9	52.4	8.9
1.40 to 1.50	12.1	26.5	64.5	12.2
1.50 to 1.70	10.6	39.8	75.1	16.1
Over 1.70	24.9	72.9	100.0	30.2

[a] Moisture-free basis.

Table 4 . . . Specific-gravity Analysis of 3½ In. to 0 Raw Slack from McKay Bed

Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Cumulative	
			Weight, Per Cent	Ash,[a] Per Cent
Under 1.30	60.1	3.0	60.1	3.0
1.30 to 1.40	14.0	8.8	74.1	4.1
1.40 to 1.50	4.7	21.4	78.8	5.1
1.50 to 1.70	5.1	38.8	83.9	7.2
Over 1.70	16.1	71.6	100.0	17.5

[a] Moisture-free basis.

The McKay bed, best known of the western Washington coals, currently is mined at Ravensdale by the Northwestern Improvement Co. and at Cumberland by the Olson Fuel Co. A coal of subbituminous A rank, the McKay has an inherent ash content of less than 3 pct and thus is by far the cleanest coal in the state. The specific-gravity analysis of raw 3½ in. slack shown in Table 4 demonstrates, however, that with the mining conditions prevailing at Ravensdale a large quantity of impurity contaminates the raw coal.

A 2-compartment Elmore jig, recently rebuilt as a steel box, handles 3½ in. slack at Ravensdale. The washed coal is sized on vibrating screens to produce the conventional egg, nut, stoker, and buckwheat sizes. At this mine, as at many other operations in the state, the sludge or slurry overflowing the washed-coal settling tank is flumed to settling ponds from which it is recovered ultimately by scraper.

Pierce County contains the only reserves of coking coal in the state, and coals from the Wilkeson-Carbonado-Fairfax district of this county were used for coke manufacture from 1880 to 1936, when the demand for coke would no longer support continued production. With the advent of the late war, however, the demand for coke was renewed, and a mine at Wilkeson was developed by the Wilkeson Products Co. to supply coal for a byproduct-coke plant at Tacoma. The washery constructed for this mine is of interest principally because of the elaborate provision made for rewashing the coal. As shown by the specific-gravity analyses in Table 5, the beds mined at Wilkeson contain an excessive quantity of heavy impurity as well as considerable bony material. Fig 2 compares these beds by means of yield-ash curves.

To deal with material of this character, the raw coal was first put through a Bradford breaker to eliminate the coarse rock and break the remainder of the material to pass 1 in. The 1-in. slack was then washed in a Vissac jig set to produce a clean refuse and a washed product still containing some impurity. This washed coal was passed over a 1 mm wedge-wire screen, the oversize was rewashed in a second Vissac jig, and the undersize was rewashed on a battery of 8 Deister-Overstrom tables.

The washed product from the second Vissac jig was screened at 4 mm, the oversize material constituting finished clean coal and the undersize material being added to the table feed. Refuse from the second jig, in reality a middling product, could be either marketed as a high-ash steam coal or crushed to

Table 5 . . . Specific-gravity Analyses of Raw Wilkeson 1 In. Slacks

Bed	Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Cumulative	
				Weight, Per Cent	Ash,[a] Per Cent
No. 1, west dip	Under 1.40	35.8	9.1	35.8	9.1
	1.40 to 1.50	13.2	23.4	49.0	13.0
	1.50 to 1.70	14.0	37.9	63.0	18.5
	Over 1.70	37.0	65.3	100.0	35.8
No. 2, west dip	Under 1.40	56.1	9.7	56.1	9.7
	1.40 to 1.50	13.0	24.3	69.1	12.4
	1.50 to 1.70	10.4	37.0	79.5	15.7
	Over 1.70	20.5	69.9	100.0	26.8
No. 3, west dip[b]	Under 1.40	47.8	9.1	47.8	9.1
	1.40 to 1.50	10.8	23.8	58.6	11.8
	1.50 to 1.70	10.8	37.8	69.4	15.9
	Over 1.70	30.6	70.6	100.0	32.6
No. 5, east dip	Under 1.40	78.5	8.1	78.5	8.1
	1.40 to 1.50	7.2	23.0	85.7	91.4
	1.50 to 1.70	5.4	36.3	91.1	10.9
	Over 1.70	8.9	65.1	100.0	15.8

[a] Moisture-free basis.
[b] ¼-in. slack.

pass ¼ in. and added to the table feed. Most of the final clean coal was produced on the tables.

This elaborate schedule of retreatment was required to produce a washed coal of 11 to 12 pct ash, corresponding to a separation at a little less than 1.50 sp gr, and demonstrates the type of washing involved if a very difficult coal must be washed at low specific gravity. The plant is not now operating and currently is offered for sale by the War Assets Administration.

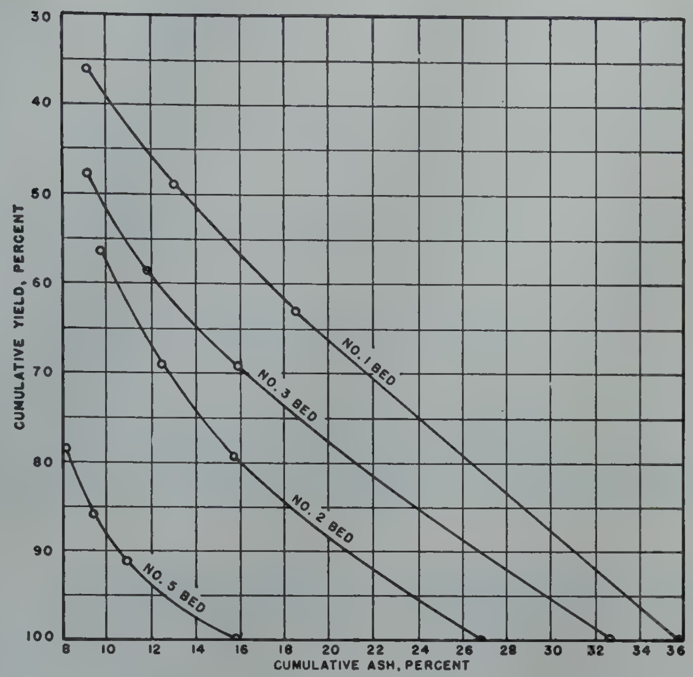

FIG 2—Yield-ash curves for Wilkeson coal beds.

Alaska

Coal production in Alaska during recent years has been divided about equally between the subbituminous coals of the Nenana field and the bituminous coals of the Matanuska Valley. The subbituminous coals, mined at Suntrana, are clean enough to be prepared by screening and some hand picking. In contrast, the coals of the Matanuska Valley contain more impurity than most so-called "dirty" coals mined in the United States and consequently must be washed to render them suitable for even the Alaska market. The proportion of washery feed rejected as refuse in Alaska averaged 33 pct in 1945, a far higher figure than that recorded for any state in the country.[1]

In 1943 the Alaska railroad built a washery at its Eska mine to prepare coal for railroad use. All coal was crushed to pass a 2½ in. bar screen and washed in a 3-cell, 1-compartment, Baum-type jig at the rate of 60 tph. Washed coal was sized into 1 in. nut and 1 in. slack on a vibrator. During the winter months coal finer than ⅜ in. was dewatered in a centrifugal drier. Detailed performance tests of this plant were made by the Bureau of Mines to determine the relationship between quality and yield of washed coal.[2] Since so little information is available on the operation of washers treating unusually dirty coal, some of the data from these tests are summarized here to illustrate the type of performance obtained.

Table 6, giving the specific-gravity analyses of the jig feed by several size fractions, indicates that 40 pct of the raw coal is impurity heavier than 1.70 sp gr and about 25 pct occurs between 1.40 and 1.70 sp gr.

Actually, the concentration of impurity in the material coarser than 1-in. is so great that this size contains little more coal than the refuse discarded at some washeries. Table 7 gives the ash contents, yields, and efficiencies obtained in a series of three tests in which the jig was operated to give a washed coal of increasingly lower ash content.

Efficiency, as the term is used in Table 7, is the ratio of the yield of washed coal to the yield of float coal of the same ash content present in the feed, expressed as a percentage. The efficiency of washing fell from 96 pct in Test 1, in which a washed coal of 20.4 pct ash was obtained, to 89 pct in Test 3, in which the washed coal contained 14.1 pct ash. These efficiencies are much lower than the values of 98 to 99 pct characterizing good washery performance, yet they

Table 6 . . . Specific-gravity Analyses of Various Sizes of Raw Coal, Eska Mine

Size	Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Cumulative Weight, Per Cent	Cumulative Ash,[a] Per Cent
Over 1 in. Weight, 28.7 pct	Under 1.30	3.2	4.6	3.2	4.6
	1.30 to 1.40	6.1	14.6	9.3	11.2
	1.40 to 1.50	12.8	25.9	22.1	19.7
	1.50 to 1.70	18.7	38.8	40.8	28.5
	Over 1.70	59.2	72.2	100.0	54.4
1 to ⅜ in. Weight, 29.1 pct	Under 1.30	15.3	4.7	15.3	4.7
	1.30 to 1.40	13.0	14.1	28.3	9.0
	1.40 to 1.50	13.8	26.5	42.1	14.7
	1.50 to 1.70	15.3	40.1	57.4	21.5
	Over 1.70	42.6	72.9	100.0	43.4
⅜ in. to 20 mesh Weight, 35.4 pct	Under 1.30	35.9	3.6	35.9	3.6
	1.30 to 1.40	17.7	12.7	53.6	6.6
	1.40 to 1.50	9.3	25.3	62.9	9.4
	1.50 to 1.70	11.5	39.4	74.4	14.0
	Over 1.70	25.6	71.2	100.0	28.7
Under 20 mesh Weight, 6.8 pct	Under 1.30	37.4	2.9	37.4	2.9
	1.30 to 1.40	17.6	9.9	55.0	5.1
	1.40 to 1.50	6.7	24.1	61.7	7.2
	1.50 to 1.70	8.4	36.2	70.1	10.7
	Over 1.70	29.9	69.7	100.0	28.3
Composite, all sizes	Under 1.30	20.7	3.8	20.7	3.8
	1.30 to 1.40	13.0	13.1	33.7	7.4
	1.40 to 1.50	11.4	25.9	45.1	12.1
	1.50 to 1.70	14.5	39.3	59.6	18.7
	Over 1.70	40.4	72.1	100.0	40.3

[a] Moisture-free basis.

Table 7 . . . Ash Content, Yield, and Efficiency, by Particle Size, Eska Mine

Size, In. and Mesh, Square Hole	Feed, Weight, Per Cent	Ash, Per Cent[a]			Yield, Per Cent		Efficiency, Per Cent
		Feed	Washed Coal	Refuse	Float Coal	Washed Coal	
Test No. 1							
Over 1	28.7	54.4	25.9	69.1	35.2	34.2	97.2
1 to ⅜	29.1	43.4	23.1	72.8	60.4	59.1	97.8
⅜ to 20	35.4	28.7	16.4	65.8	79.3	75.3	95.0
Under 20	6.8	28.3	21.4	55.7	89.4	80.0	89.5
Weighted average	100.0	40.3	20.4	69.0	61.8	59.1	96.1
Test No. 2							
Over 1	26.9	53.7	20.4	64.9	27.6	25.1	90.9
1 to ⅜	29.5	41.2	18.4	66.7	55.2	52.8	95.7
⅜ to 20	35.6	28.0	14.2	62.8	76.2	71.6	94.0
Under 20	8.0	27.1	19.9	53.5	89.0	78.4	88.1
Weighted average	100.0	38.8	16.9	64.6	58.0	54.1	93.2
Test No. 3							
Over 1	28.6	51.5	16.9	59.3	21.6	18.4	85.2
1 to ⅜	29.0	43.3	15.6	61.1	42.6	39.3	92.3
⅜ to 20	35.0	30.6	12.0	58.0	65.8	59.7	90.7
Under 20	7.4	26.0	16.6	48.4	84.5	69.8	82.6
Weighted average	100.0	39.9	14.1	59.1	47.8	42.7	89.0

[a] Moisture-free basis.

Table 8 . . . Specific-gravity Analyses of Raw Coal from Southport Mine

Size, In. and Mesh	Specific Gravity	Weight, Per Cent	Ash,[a] Per Cent	Cumulative	
				Weight, Per Cent	Ash,[a] Per Cent
2 to 1½ in.	Under 1.40	73.2	9.8	73.2	9.8
Weight, 10.9 pct	1.40 to 1.50	10.6	22.9	83.8	11.5
Ash, 22.4 pct[a]	1.50 to 1.70	2.8	39.3	86.6	12.4
	Over 1.70	13.4	87.8	100.0	22.4
1½ to ¾ in.	Under 1.40	72.4	9.8	72.4	9.8
Weight, 21.3 pct	1.40 to 1.50	11.2	22.1	83.7	11.4
Ash, 22.1 pct[a]	1.50 to 1.70	3.4	40.3	87.1	12.6
	Over 1.70	13.0	85.9	100.0	22.1
¾ to ⅜ in.	Under 1.40	72.4	9.4	75.4	9.4
Weight, 23.7 pct	1.40 to 1.50	9.9	21.2	85.3	10.8
Ash, 20.1 pct[a]	1.50 to 1.70	3.7	39.0	89.0	11.9
	Over 1.70	11.0	86.3	100.0	20.1
⅜ in. to 20 mesh	Under 1.40	72.4	8.2	72.4	8.2
Weight, 36.2 pct	1.40 to 1.50	10.1	18.4	82.6	9.4
Ash, 19.3 pct[a]	1.50 to 1.70	3.6	33.6	86.2	10.5
	Over 1.70	13.9	74.0	100.0	19.3
Under 20 mesh	Under 1.40	49.0	8.0	49.0	8.0
Weight, 7.9 pct	1.40 to 1.50	11.0	18.4	60.0	9.9
Ash, 34.7 pct[a]	1.50 to 1.70	7.1	32.9	67.1	12.3
	Over 1.70	32.9	80.3	100.0	34.7
Composite	Under 1.40	71.3	9.0	71.3	9.0
Weight, 100.0 pct	1.40 to 1.50	10.4	20.4	81.7	10.5
Ash, 21.6 pct[a]	1.50 to 1.70	3.8	36.5	85.5	11.6
	Over 1.70	14.5	81.0	100.0	21.7

[a] Moisture-free basis.

do not represent particularly faulty jig operation. The percentage of coal in the refuse product was no higher than that obtained in normal jig operation, but the abnormal quantity of refuse rejected, ranging from 40.9 pct in Test 1 to 57.3 pct in Test 3, made this nominal percentage of coal constitute a serious washery loss.

With the conversion of the Alaska railroad from coal to oil, the Eska mine was closed and put in standby condition.

At the mine of the Evan Jones Coal Co., a few miles from Eska, the coal contains considerably less impurity but still constitutes a washing problem of more than usual difficulty. The egg, nut, and ¾ in. slack sizes are washed separately in a single Forrester-type jig of about 30 tph capacity. As might be expected, manually controlled equipment of this type is not particularly satisfactory for treating a difficult coal, even in the hands of an experienced, competent operator. Fluctuations in the quality of the washed coal occur, and the refuse product may contain more than the usual amount of coal.

Oregon

Coal occurs at a number of locations in western Oregon, but mining on a substantial scale has been limited to the Coos Bay field, on the southern coast of Oregon. Coos Bay coal was first washed at the Beaver Hill mine, which supplied fuel for the Southern Pacific Railroad in the early 1900's. The present Southport mine of the Coast Fuel Corp., started during the closing years of the recent war, also required a washing plant to remove the interbedded clay of the Beaver Hill bed. Located 5 miles south of Coos Bay, this plant incorporates provision for screening the run-of-mine product at about 4 in. size, hand-picking the lump, and optionally crushing the clean lump to join the screenings as feed for a Forrester-type jig. Washed coal is sized on a vibrating screen into the conventional egg, nut, and stoker sizes, with the material finer than ¼ in. screened from the stoker coal and flumed to a storage pond for possible future utilization.

Table 8 shows a specific-gravity analysis, by size fractions, of raw coal from this mine. The material represented was run-of-mine coal from which 5 pct of clean rock had been removed by hand picking. The remainder of the material was crushed to pass 2 in. square-hole. With less than 15 pct of impurity heavier than 1.70 sp gr and little material of intermediate density, the Southport coal is cleaner than most Washington or Alaska coals. A washed product of 12 to 13 pct ash, moisture-free basis, can be produced readily from this coal. The only difficulty encountered arises from the character of the clay that constitutes the principal impurity. This clay disintegrates readily in water and therefore tends to contaminate the finer sizes of coal, from which its removal is more difficult; the clay would also build up in a closed-circuit water system.

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A Study of Coal Classification and Its Application to the Coking Properties of Coal

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The fact that coal is a complex organic material and heterogeneous in composition has made its study extremely difficult, particularly in regard to obtaining a fundamental concept of the processes involved in the formation of coke. It has been stated[1] recently that coals are natural polymers and that in a complex reaction such as occurs during carbonization the differences between coals are quantitative and not qualitative. Such being the case, it would be expected, therefore, that a series of coals subjected to exactly the same conditions of coking would produce results that should be related quantitatively to the chemical analyses of the individual coals. In the past, however, this fact was not generally recognized and consequently many empirical test methods were devised for testing coals. The interpretation of the results obtained depended upon comparing a set of data with other data on coals of known characteristics. The testing procedure had to be conducted in precisely the same manner each time and not the least requirement in making the interpretation was experience. The test methods varied from those utilizing a gram of coal to the plant-scale tests in which one or more ovens were charged with the test coal. The aim had been principally to find some simple, inexpensive means by which the coke-making and other properties of coal could be predicted. It is not necessary to emphasize the importance of these endeavors, for anyone acquainted with the problems involved in the selection of coals for making coke is well aware of the time and expense involved in making a series of definitive tests.

Several attempts have been made to derive formulas for such predictions. The relations of both ultimate and proximate analyses to coke physical properties and byproduct yields have been studied. Lowry and co-workers[1] correlated statistically the U. S. Bureau of Mines-American Gas Association assay test results with coal analyses, using 90 coals and coal blends. Parry,[2] Spooner,[3] and Gabinskii and Krym[4] made similar correlations. Lowry concluded that there were definite relationships between the proximate analyses of the coals and the yields and properties of the products. Although this statistical study was made on test results from carbonizing coal in a cylindrical steel retort, the conclusions are important and provided one of the first steps in the simplification of this complex subject. Studies should be made of other references on this subject just cited.

Studies of the coking process reported in the literature frequently were confined to a small number of coals, and very few have included coals from various ranks. It would seem logical, however, that a program for studying factors involved in the formation of coke should be based on a number of coals selected from each of the various classes or types of coking coal. The possible use of the scheme of classifi-

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[1] References are at the end of the paper.

Table 1 . . . ASTM Standard Classification of Coals by Rank

Class	Group	Limits of Fixed Carbon or British Thermal Units, Mineral-matter-free Basis[a]	Requisite Physical Properties
I. Anthracitic	1. Meta-anthracite	Dry FC, 98 pct or more (dry VM, 2 pct or less)	
	2. Anthracite	Dry FC, 92 pct or more and less than 98 pct (dry VM, 8 pct or less and more than 2 pct)	
	3. Semianthracite	Dry FC, 86 pct or more and less than 92 pct (dry VM, 14 pct or less and more than 8 pct)	Nonagglomerating
II. Bituminous	1. Low-volatile bituminous coal	Dry FC, 78 pct or more and less than 86 pct (dry VM, 22 pct or less and more than 14 pct)	
	2. Medium-volatile bituminous coal	Dry FC, 69 pct or more and less than 78 pct (dry VM, 31 pct or less and more than 22 pct)	
	3. High-volatile A bituminous coal	Dry FC, less than 69 pct (dry VM, more than 31 pct); and moist Btu 14,000 or more	
	4. High-volatile B bituminous coal	Moist Btu, 13,000 or more and less than 14,000	
	5. High-volatile C bituminous coal	Moist Btu, 11,000 or more and less than 13,000	Either agglomerating or nonweathering
III. Subbituminous	1. Subbituminous A coal	Moist Btu, 11,000 or more and less than 13,000	Both weathering and nonagglomerating
	2. Subbituminous B coal	Moist Btu, 9,500 or more and less than 11,000	
	3. Subbituminous C coal	Moist Btu, 8,300 or more and less than 9,500	
IV. Lignitic	1. Lignite	Moist Btu, less than 8,300	Consolidated
	2. Brown coal	Moist Btu, less than 8,300	Unconsolidated

[a] FC, fixed carbon; VM, volatile matter; Btu, British thermal units.

cation of coal by rank based on chemical analyses as a tool in the study of the behavior of coking coal was emphasized by the authors in 1942.[5] It was said in part at that time, " . . . a good understanding of the principles involved in setting up the standard method of classification may lead to a better appreciation of the causes of changes in coke characteristics."

There is sufficient evidence obtained from past experience to indicate that characteristics of coke produced from a given coal are in some measure related to the rank of coal. Broadly speaking, the high-volatile C bituminous coals make very small fragile cokes if the coals coke at all; whereas the high-volatile A coals produce much better cokes. Although the low-volatile bituminous coals produce strong cokes, they cannot be carbonized unblended in byproduct coke ovens because they exert excessive pressures against the oven walls. While these generalities seem to be obvious, the question immediately arises as to whether or not small changes in rank produce quantitative differences in coking behavior.

The purpose of this study is to present a discussion of coal classification by rank with the introduction of a new scheme developed by the authors. Some preliminary results of correlating coking behavior with coal rank will be presented.

Classification of Coal

The fact that coal is a natural product derived from a wide variety of vegetable matter that has been altered by bacterial action, heat, pressure, and time gives rise to a continuous series in which there are no obvious boundaries between different types. A method by which coals could be classified was sought by a large number of investigators over a period of more than a hundred years. During that time a very large number of schemes were proposed. It would be impossible to present even a cursory account of the literature on classification. The reader is referred to an excellent review of the subject as written by Dr. H. J. Rose.[6]

The methods devised in the past may be divided into three categories: (1) methods based on proximate analyses, (2) methods based on ultimate analyses, and (3) methods based on both proximate and ultimate analyses. The heating value of the coal is included in one or another of the schemes of classification and this property is an important factor in the standard procedure for classification of coal by rank.

ASTM CLASSIFICATION BY RANK

Through the joint efforts of the American Standards Association, the American Society for Testing Materials, and the American Institute of Mining and Metallurgical Engineers, the present American standard for classification of coal was established about ten years ago. This development of a classification scheme by the unified efforts of a large group of scientists and technologists of acknowledged reputation was one of the outstanding accomplishments in coal technology. Despite the widespread interest and activity that extended over a period of more than ten years in which the method was developed, it is surprising that little attention has been paid to applying the method to coal carbonization.

The basis for the scheme of classification is the proximate analysis and the heating value of the coal. The fixed carbon of the coal is calculated to the dry, mineral-matter-free basis, and the heating value in Btu per pound is calculated to the moist, mineral-matter-free basis. Table 1 presents the details of the classification as designated by the ASTM. This scheme places all of the coking coals in the bituminous class. The ASTM Standards[7] should be consulted for the various details of the specifications. Definitions of agglomerating, nonagglomerating, weathering, and nonweathering are given because these factors are used to separate the bituminous class from the anthracitic on the one side, and

from the subbituminous on the other. Each of the groups merges into the adjoining one and the boundary lines between the groups are arbitrary limits established by agreement.

Fig 1 is a plot of more than 300 coal analyses of all ranks from meta-anthracite to lignite. This is a composite of the various charts shown in *R. I.* No. 3296 published by the U. S. Bureau of Mines with the exception that the scale of the abscissa is reversed. The coal analyses used in the construction of these classification charts and also including Fig 2, 4, 5, and 6 were selected by the U. S. Bureau of Mines from their various bulletins and technical papers on the analyses of United States coals. Although the method of classification that was finally adopted specifies the use of "bed moisture" for the correction of the heating value of coal to moist, mineral-matter-free basis, all analyses used in *R. I.* No. 3296 are on the "as-received" basis and in consequence the charts developed in this study of coal classification are also on the same basis.

More detailed attention must be given to the terms "mineral matter" and "moisture" in this discussion, since a thorough understanding of these terms is not only necessary in consideration of the standard classification but also of the modified classification to be described in this paper. Mineral matter is the ash material as it exists in the original coal. During the determination of ash, the water of hydration of the slate, and so on, is driven off, and the pyrite which is represented by the chemical formula FeS_2 is converted to Fe_2O_3. The ash that is weighed in the determination is consequently of different composition and weight than the original mineral matter. Parr and Wheeler[8] developed empirical formulas by which ash as weighed could be converted to the original mineral matter. Coal corrected to the mineral-matter-free basis was called "unit coal" by those authors, but the ASTM has preferred to designate this correction as "mineral-matter-free." The following are the formulas developed by Parr and Wheeler with some later amplification by these authors:

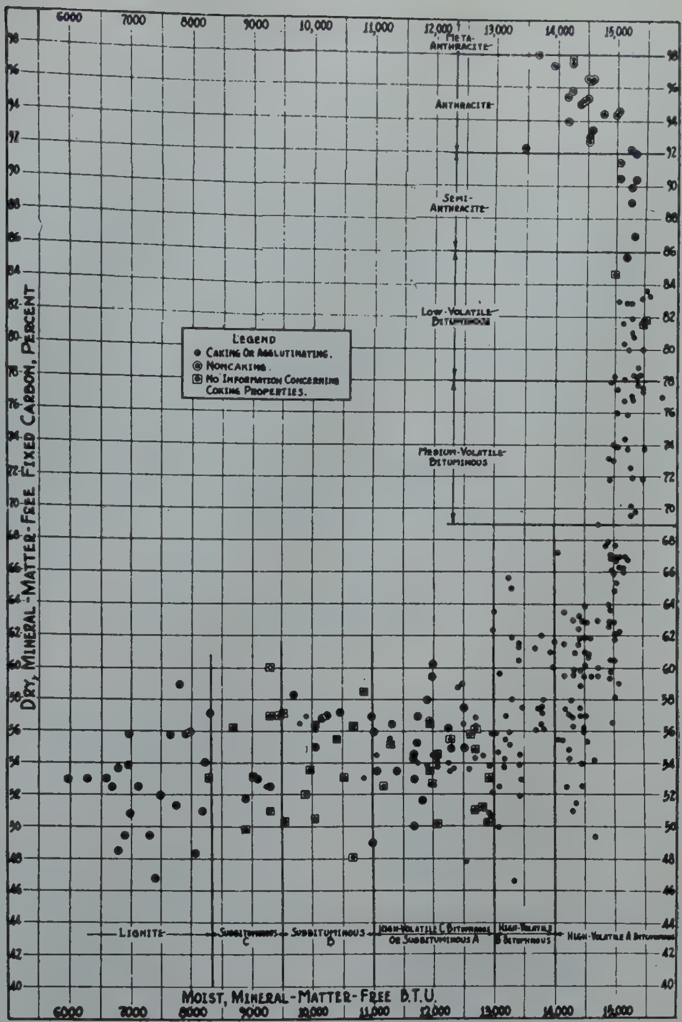

FIG 1—Classification of typical United States coals. (U.S. Bureau of Mines *R.I.* 3296.)

PARR FORMULAS

Dry, Mm-free FC

$$= \frac{FC - 0.15S}{100 - (M + 1.08A + 0.55S)} \times 100$$

Dry, Mm-free VM
$= 100 -$ Dry, Mm-free FC

Moist, Mm-free Btu

$$= \frac{Btu - 50S}{100 - (1.08A + 0.55S)} \times 100$$

Where: Mm = Mineral matter
FC = Fixed Carbon
VM = Volatile Matter
S = Sulphur
M = Moisture
A = Ash
Btu = British thermal units

Simplified formulas are given below called "Approximation Formulas." The Parr formulas are accepted in all cases where litigation is involved.

APPROXIMATION FORMULAS

Dry, Mm-free FC

$$= \frac{FC}{100 - (M + 1.1A + 0.1S)} \times 100$$

Dry, Mm-free VM
$= 100 -$ Dry, Mm-free FC

Moist, Mm-free Btu

$$= \frac{Btu}{100 - (1.1A + 0.1S)} \times 100$$

The moisture content of the coal used in the scheme of classification is the bed moisture. Because this factor is so important and possibly may be misunderstood, the following is a verbatim copy from the ASTM standard[7] in which the "moist" basis is specified:

(*d*) In case the coal is likely to be classified on the "moist" basis, that is, containing the natural bed moisture, the samples shall be taken at freshly exposed faces, which are free from visible surface moisture if possible. Samples of low-rank coals which appear dry at the time of collection frequently give off moisture which condenses on the inner surface of the sample containers, before they are opened for analysis. In the case of coals which were free from visible surface moisture when sampled, but which show moisture on the inner surface of the container when opened, both the container and the coal shall be weighed before and after

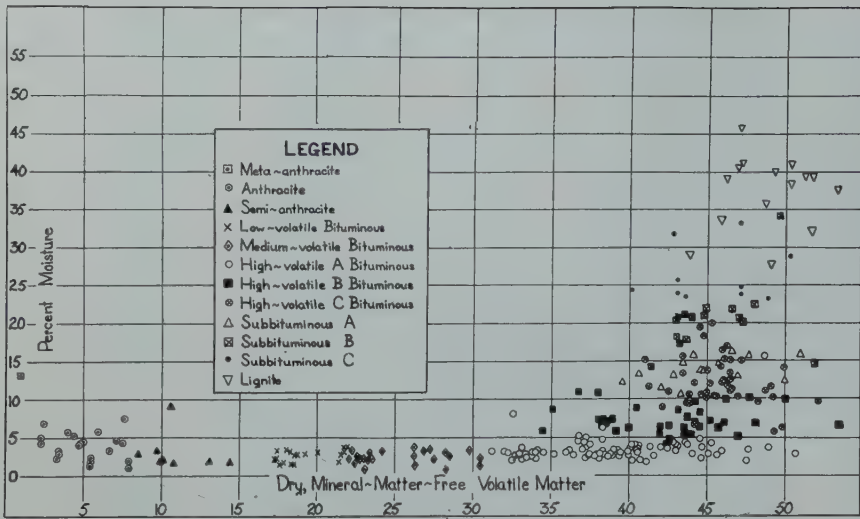

FIG 2—Variation of moisture content of coal with rank. (Data from U.S. Bureau of Mines *R.I.* 3296 R.)

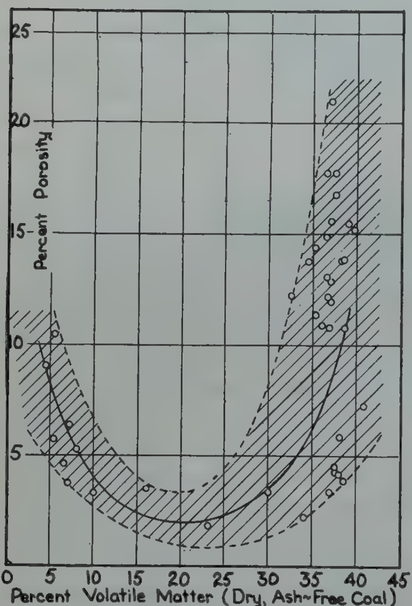

FIG 3—Variation of coal porosity with volatile matter. (Data from The Ultra-fine Structure of Coals and Coke p 48 BCURA 1944.)

air-drying and the total loss in weight shall be reported as air-drying loss.

(*e*) If it is impossible to sample the coal without including visible surface moisture, and the coal is likely to be classified on the "moist" basis, the sampler shall include the following statement in the description: "Sample contains surface moisture." Samples so marked shall not be used for classification on a moist basis unless brought to a standard condition of moisture equilibrium at 30 C. in a vacuum desiccator containing a saturated solution of potassium sulfate (97 per cent humidity) as suggested by Stansfield and Gilbart.[5] Analyses of such wet samples which have been treated in this manner shall be designated as "wet samples equilibrated at 30 C. and 97 per cent humidity.

"[5] Edgar Stansfield and K. C. Gilbart, "Moisture Determination for Coal Classification," Transactions, Am. Inst. Mining and Metallurgical Engrs., Coal Division, p. 125 (1932)."

From the above, it is evident that the bed moisture is the amount of water held within the particles of coal. It is not water on the surface that might come from overlying strata; it is not water held in chemical combination with either the coal substance or the mineral matter; it is water that can be expelled at 105°C, and it is water that is held within the pore spaces. The as-received basis on which coal analyses are frequently reported means that the analysis was made on coal in the condition in which it arrived at its destination. The moisture content of the coal may or may not be the same as it was before mining, and such may be true for the other determinations. Coal sampled from the "face" in the mine may be wet due to ground water and in that case the moisture content of the sample will be higher than the bed moisture. Thus, moisture in coal as-received is not necessarily the same as bed moisture but in the case of samples taken at a fresh face in the mine, the moisture content may be very nearly the same or identical with bed moisture. As stated above, the analyses used in this study of classification of coal were made on the as-received basis. The samples for these analyses were all taken from the face. Apparently there is not sufficient information available to judge the condition of these coals at the sampling face in regard to moisture. In any event the moisture content as given may be too high in many cases but in no case is it too low. It is believed that in general the moisture content of the coals used is a fair approximation of the bed moisture with the possible exception of the coals in the anthracite group.

Although the moisture content is much more important in the low-rank and noncoking coals and in that range is a significant factor in delineating coals, it also has some importance in the anthracite and meta-anthracite ranks. Variations in moisture content of coals of varying rank are shown in Fig 2 where it is seen that this constituent of coal is practically constant at 2 to 4 pct for semianthracite, low-volatile, medium-volatile, and most high-volatile A coals; but increases rapidly through the high-volatile B and C, subbituminous, and lignite coals, and less rapidly through the anthracite and meta-anthracite coals. That moisture is a fundamental property of the coal is indicated in Fig 3[9] where the relation between coal porosity and volatile matter is illustrated. Comparing Fig 2 with Fig 3, there is evidence of similarity of curves with the minimum in both cases occurring at about 20 pct dry, ash-free or mineral-matter-free volatile matter. Variation of heating value on the dry, mineral-matter-free basis with volatile matter is shown in Fig 4 and it is to be noted that it is also similar to Fig 2 and 3 although it bears an inverted relation to them. The maximum heating value occurs at about 20 pct dry, mineral-matter-free volatile matter. The source of data for Fig 2 and 4 is the same as for the classification curve in Fig 1.

MODIFICATION OF ASTM CLASSIFICATION

The study of the relation of coking properties to coal rank that will be described, resulted from an effort to use the standard method of classification for the prediction of the coking properties of coal. Many coals whose coking characteristics were known to the authors were superimposed on a chart similar to Fig 1. An examination of this plot indicated that in the high-volatile region of coals, for instance, there were cases where different coals having the same fixed carbon content on the dry, mineral-matter-free basis had quite different heating values on

the moist, mineral-matter-free basis as well as different coking characteristics. For example, some of the Illinois coals were found to have substantially the same fixed carbon as an Elkhorn seam and a No. 2 Gas seam coal, while two Pittsburgh seam coals from northern West Virginia had lower fixed carbon values than the Illinois coals, yet the Illinois coals were in the high-volatile B group and the others were of the high-volatile A rank. An examination of the coking behavior of these coals produced results that were equally anomalous. In other words, the standard classification method did not lend itself readily to the correlation of the coking properties of different coals.

The efforts of the authors, therefore, turned toward developing a method of classification that would be more adaptive to such correlation. A number of methods were studied and finally the system shown in Fig 5 was obtained. In this scheme of classification the ratio of the heating value of the coal in Btu per pound to percentage of volatile matter is plotted along the vertical logarithmic scale and percentage of volatile matter is plotted along the horizontal arithmetic scale. Both the heating value and volatile matter are on the mineral-matter-free basis and calculated with the use of the approximation formulas given above. The former is also on the moist basis while the volatile matter is on the dry basis. In Fig 5, nearly all the coals of Fig 1 were used so that the two methods are comparable.

Close examination of this plot reveals that the points representing the various coals fall very nearly on the line as shown from the anthracite group on the left and proceeding through the high-volatile A group on the right. The boundaries between the groups are distinct and according to the ASTM specifications. Below the high-volatile A group, the high-volatile B coals will be found to arrange themselves in a line just below the high-volatile A; the high-volatile C coals are found just below the high-volatile B coals, and so on down through the lignites.

While this plot achieved in a large measure the purpose of the study, the scattering of the coals below the high-volatile A group, even though fairly regular, was not entirely satisfactory. In seeking an explanation of the scattering, it appeared evident that the moisture content of the low-rank coals was accountable. By adding the moisture content to the dry, mineral-

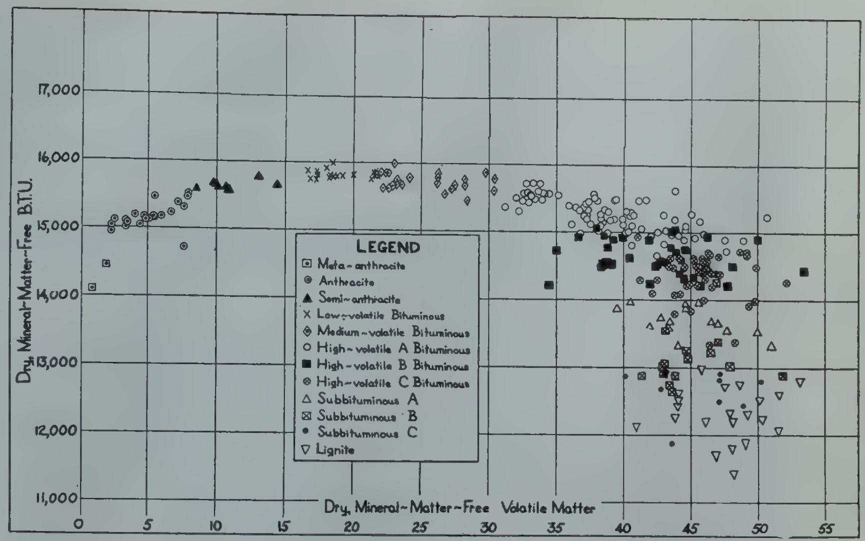

FIG 4—Variation of calorific value of coal with rank. (Data from U.S. Bureau of Mines *R.I.* 3296 R.)

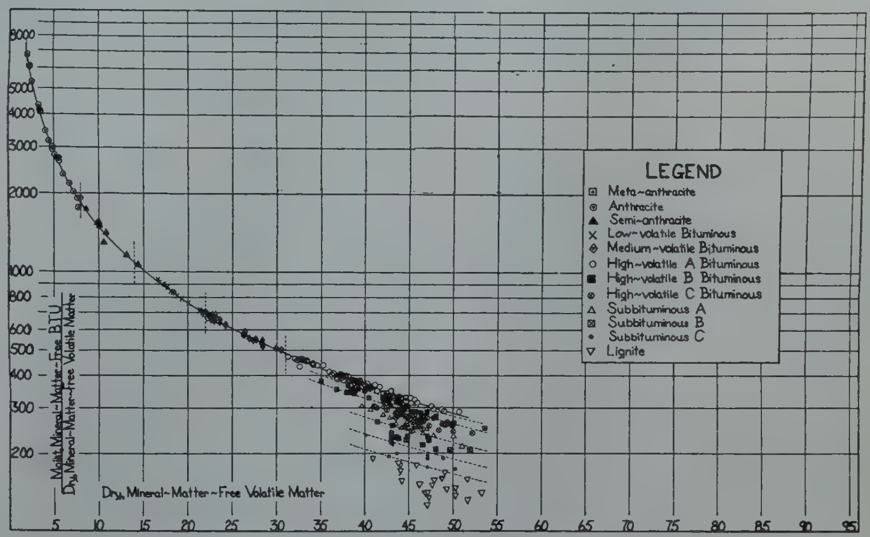

FIG 5—Classification of typical coals of the United States. (Data from U.S. Bureau of Mines *R.I.* 3296 R.)

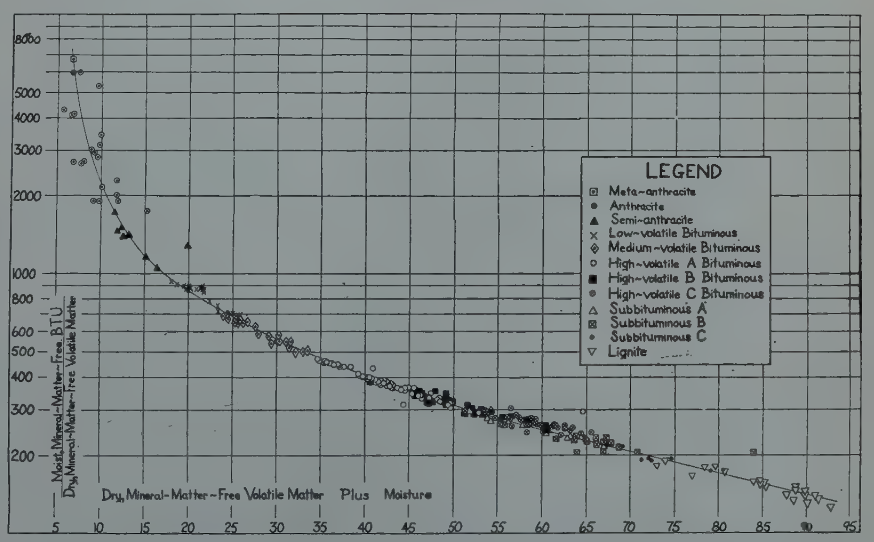

FIG 6—Classification of typical coals of the United States. (Data from U.S. Bureau of Mines *R.I.* 3296.)

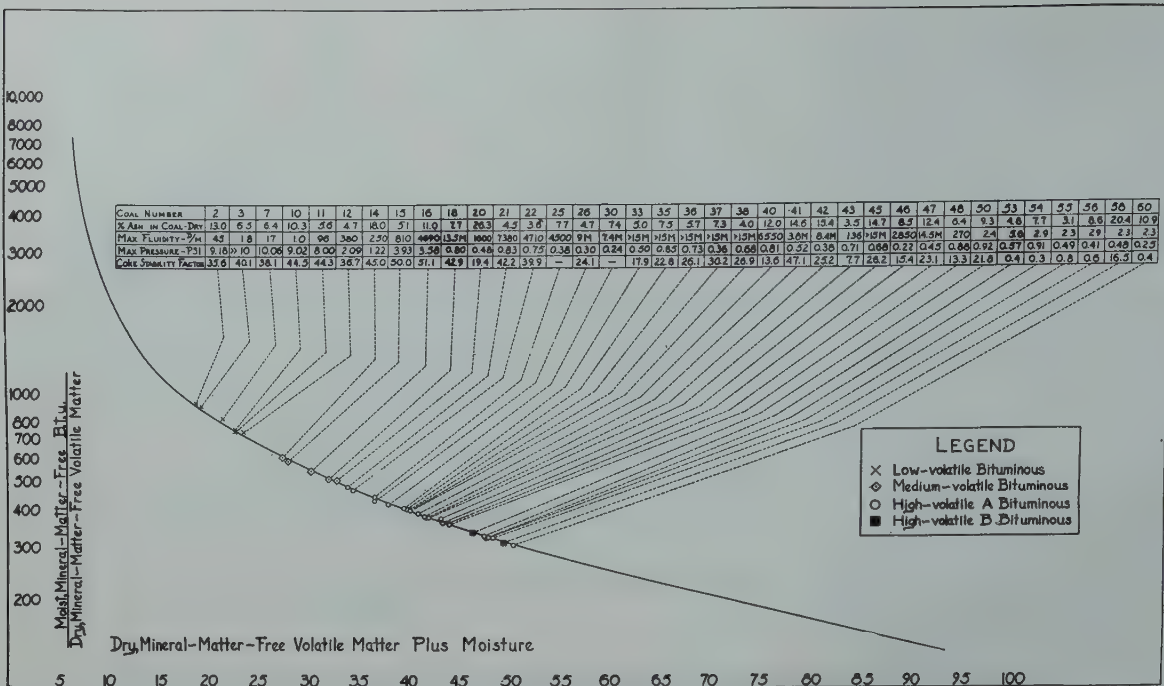

FIG 7—Relation of coal and coke properties to location of coals on classification curve.

matter-free volatile matter and using the same values on the vertical scale, the plot in Fig 6 resulted. With this change the coals below the high-volatile A group are brought into line, although the coals in the semianthracite and anthracite groups are now a little scattered.

A general examination of this modified classification scheme revealed that the coals appeared to be aligned in order of their determined coking behavior based on the experience in testing coals and coal blends in the movable-wall oven as well as in full-scale oven coking tests. Fig 7 has been constructed to show the classification curve in the bituminous group and on this curve have been plotted 36 coals whose coking properties were determined in the movable-wall oven. The data show the Gieseler test values, pressures developed, tumbler test stability factors of the coke, and ash contents of the coals charged. All of the coals used on this chart are unblended coals. Although it is evident that there are a number of cases where the sequence of values of the coking behavior is not continuous in traversing the curve of coal rank, there is a very definite trend. A number of the cases where coals appear to be out of sequence can possibly be explained by their ash content. In other cases, such exceptions might be explained by variation in petrographic constituents and undoubtedly the inaccuracies of the testing methods available can account for other variations. It should be said, however, that the testing of the coals and the cokes produced was done by the standard methods. Every effort was made to conduct the tests as accurately as possible. However, it must be acknowledged that present procedures for measuring the physical properties of coke require further refinement before satisfactory accuracy can be obtained.

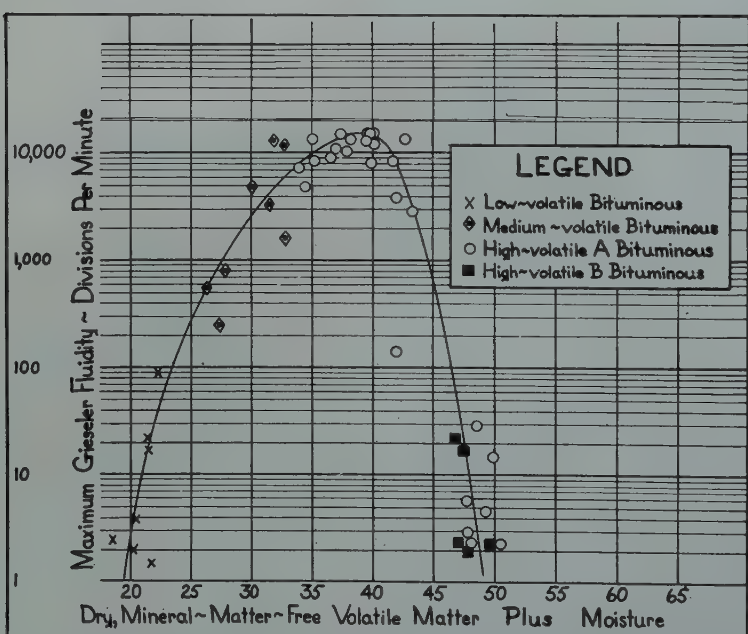

FIG 8—Relation of maximum Gieseler fluidity to coal rank.

Correlation of Coking Behavior with Coal Rank

GIESELER MAXIMUM FLUIDITY

A more detailed study was therefore undertaken to establish any relation between the classification scheme developed and the coking behavior of

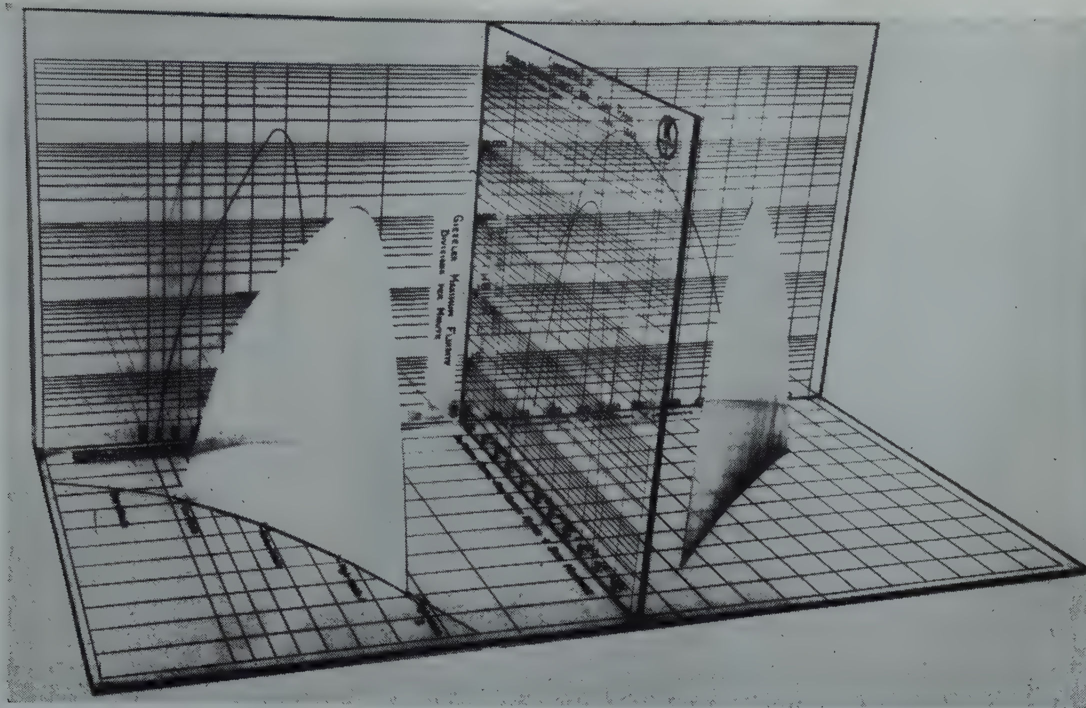

FIG 9—Correlation of Gieseler maximum fluidity and coal rank.

coals. The Gieseler test[10] for the plastic properties of coals has been used by Koppers Company, Inc., Research Department for about ten years and the data thus accumulated were first applied. Fig 8 presents the relation of the maximum Gieseler fluidity of coals to coal rank as indicated by the dry, mineral-matter-free volatile matter plus moisture. Only the coking coals are shown since the others do not exhibit fluid properties when heated. A relatively good correlation is evident with the points falling on or near a hump-shaped curve. Low-volatile coals exhibiting low fluidities are found at the lower left end of the curve. Further up the curve to the right are found medium-volatile coals and at or near the peak of the curve are found the highly-fluid high-volatile A coals. Along the downward portion of the curve to the right are found less fluid high-volatile A and B coals or coals containing higher amounts of oxygen. The peak of the curve is found to correspond to approximately 40 pct dry, mineral-matter-free volatile matter plus moisture. In other correlations to be shown later, it will be seen that the peaks also occur at about this same value.

Fig 9 is a three-dimensional plot of Gieseler data in two quadrants. In the quadrant on the left, the plane of the base contains the classification curve based on the moist, mineral-matter-free Btu divided by the dry, mineral-matter-free volatile matter plotted against the dry, mineral-matter-free volatile matter plus moisture. The vertical axis is the Gieseler maximum fluidity in divisions per minute. Data for the coals so far tested of various ranks lie on or near the plane surface shown in the figure. It will be evident

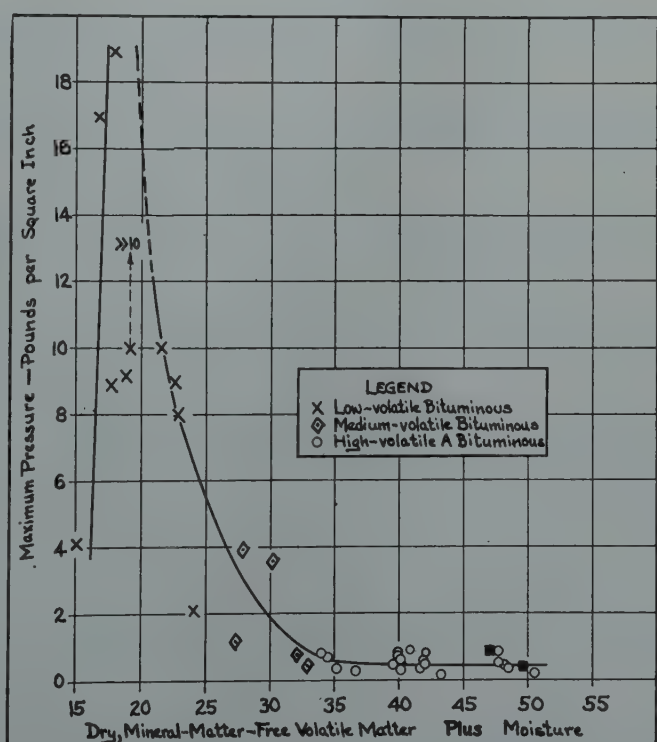

FIG 10—Relation of movable-wall oven maximum pressures to coal rank.

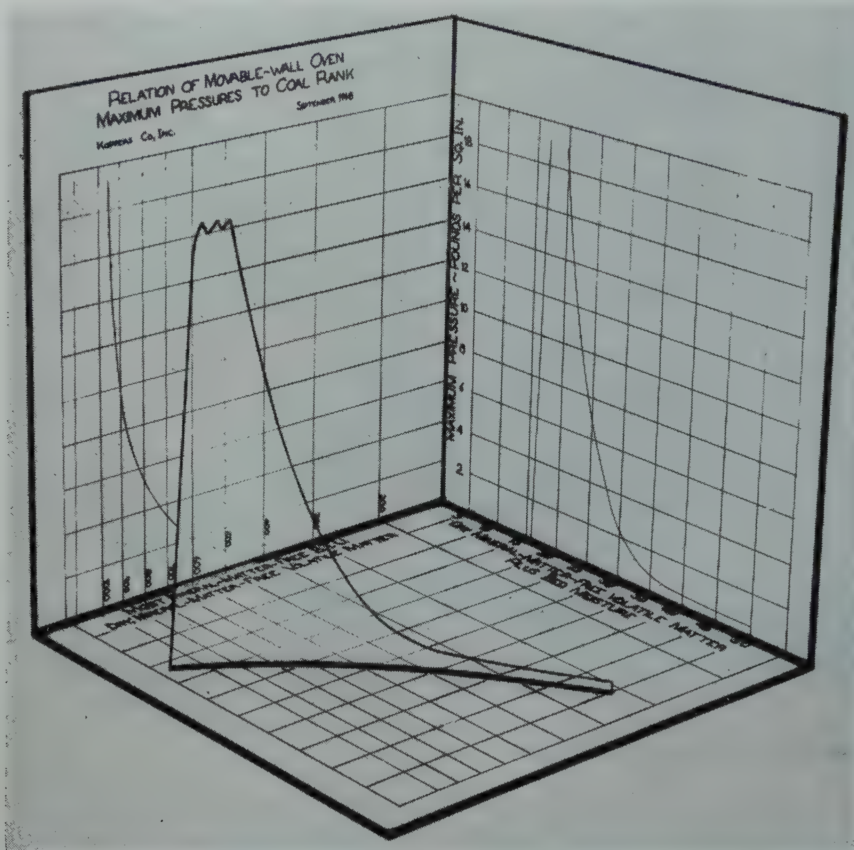

FIG 11—Relation of movable-wall oven maximum pressures to coal rank.

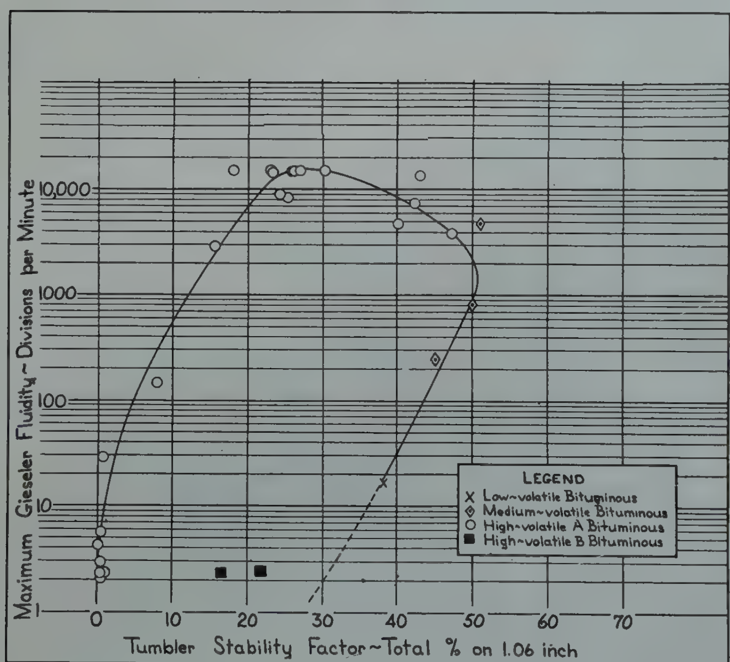

FIG 12—Relation of tumbler stability factors of movable-wall oven cokes to maximum Gieseler fluidities.

that the curve of Fig 8 is a projection of this plane surface on the right hand vertical coordinate plane. The quadrant to the right contains a base having the coordinates of dry, mineral-matter-free volatile matter plus moisture and temperature in degrees Centigrade of the maximum fluidity. The vertical axis as in the left hand quadrant is the maximum fluidity in divisions per minute. A similar plane surface erected in this right hand quadrant shows the relation of these three last-named variables. This relation will be useful in a further study of the work of Marquard[11] in which he stated that coals of different softening temperatures when blended before coking produce poorer coke than when coals of substantially the same softening temperatures are blended. While Marquard's work referred specifically to various high-volatile coals, it should now be possible to study this idea with blends of any kinds of coal. It should be noted that the coals having the highest maximum fluidity attain this condition at approximately 435°C.

The value of the Gieseler plastometer lies in the fact that significant information can be obtained from a small sample (4 to 5 g) of coal which together with the analysis heretofore had yielded an estimate of the coking behavior of the coal. In the past, the appraisal of unknown coals has been made almost entirely by relatively large-scale tests. Box tests, for example, requiring about 60 lb of coal were considered to be about the smallest test procedure that would give an indication of the type of coke to be expected. After box tests, a full-scale oven test was frequently required. The Gieseler test has not eliminated this procedure but it has been the source of information from which a more intelligent and practical approach to the selection of coals could be made. The fact that the maximum fluidity can be correlated with values obtained from proximate analyses may lead ultimately to the use of proximate analyses as a guide to the selection of coals. However, there is still much to be learned from the Gieseler test about the mechanism of coke formation.

PRESSURES DEVELOPED BY COAL

The next correlation attempted was with the maximum pressures developed by coals in the movable-wall oven. Fig 10 contains data on unblended coals, and while the information at hand is far from complete, there is shown a very definite trend in this factor with the rank of coal. All of the high-volatile A coals shown exerted a pressure of less than 1 psi. Several of the medium-volatile coals exerted more than 1 psi and are shown to go up as high as 4 lb as the rank increased

toward the low-volatile group. Two of these coals contained a high percentage of ash and the latter acting as an inert had reduced the pressure-developing properties of the coals resulting in the points falling below the curve. The low-volatile coals exerted higher and higher pressures as the rank increased and finally reached a maximum whose value is still undetermined. This is due, in part, to the inability to measure these high pressures under the same conditions that are used for measuring the pressures of the lower rank coals. As the rank increased still further the pressure dropped off. Several English coals are shown in this area and one with a dry, mineral-matter-free volatile matter plus moisture of about 15 pct was found to exert a pressure of just 4 psi. This behavior with changing rank appears to have some reason in fact, for as the rank increases to the semi-anthracite group and the fluidity decreases, such coals can conceivably produce little or no pressures. Much more work needs to be done in this area as well as in medium-volatile group near the low-volatile boundary. No satisfactory relation has yet been found that will indicate the pressures exerted by blends of coal.

Fig 11 is a three-dimensional plot of pressures developed in the movable-wall oven. The plane of the base contains the classification curve and plotted vertically is the pressure developed in pounds per square inch. It is evident that the preceding figure (Fig 10) is really a projection of the curved surface rising out of the base projected on the right hand coordinate plane.

MAXIMUM FLUIDITY AND STABILITY FACTOR

Fig 12 presents the relation between the Gieseler maximum fluidity of unblended coals and the tumbler test stability factors of the resultant cokes. Although the data are not complete, there is sufficient information on hand to show that the curve fixes the approximate relationship. Low-volatile coals having a low maximum fluidity produce coke of higher stability factor than the high-volatile coals having substantially the same maximum fluidity. The medium-volatile coals and the high-volatile A coals near the medium-volatile group produce coke of the highest stability factor. Coals of the highest maximum fluidity produce cokes of intermediate stability factor.

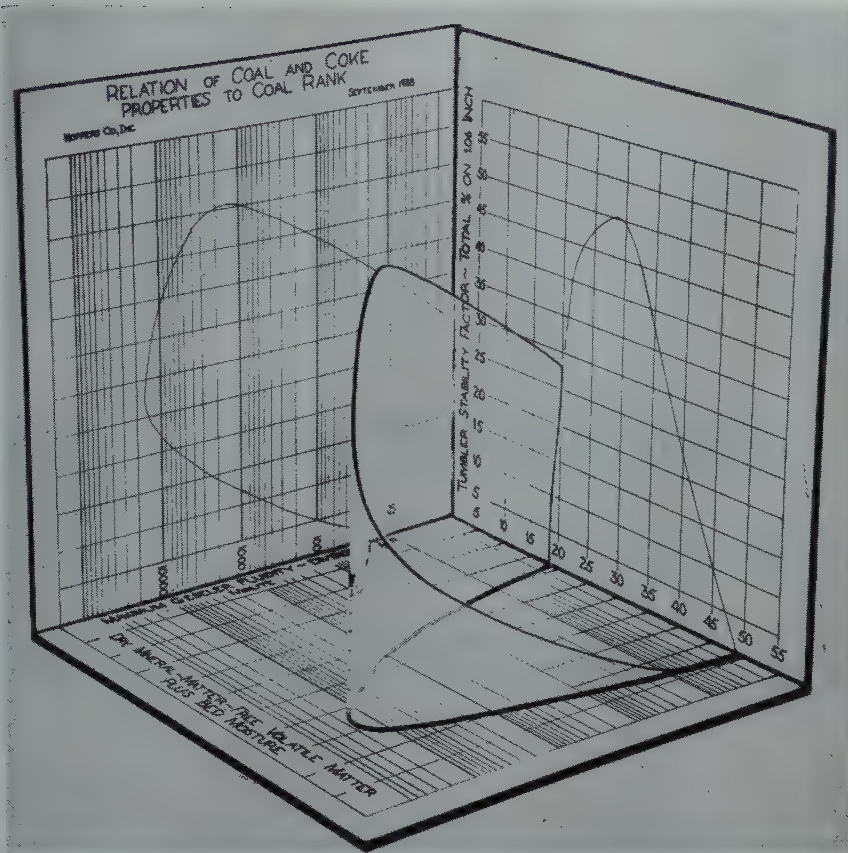

FIG 13—Relation of coal and coke properties to coal rank.

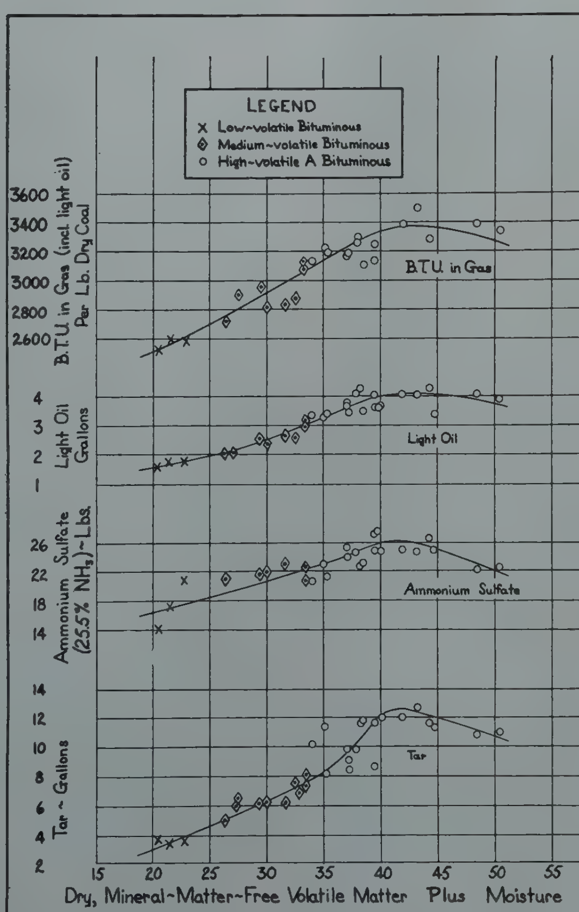

FIG 14—Relation of byproduct yields to coal rank.

The high-volatile A coals having a low maximum fluidity have the lowest stability factor. The two high-volatile B coals shown on the chart do not fit into the curve as shown. They appear to have a stability factor corresponding to the same values as the high-volatile A coals of the highest maximum fluidity. The reasons for the behavior of these two coals is still obscure but in finding the reasons for such anomalies an explanation of the overall coking behavior may be forthcoming.

Here again the curve shown in Fig 12 is the projection of the plane surface as shown in Fig 13. This three-dimensional plot results from plotting the tumbler test stability factor of the cokes with the dry, mineral-matter-free volatile matter plus moisture as the base and the Gieseler maximum fluidity as the vertical axis. Data thus plotted lie along the top of the curved plane illustrated.

While the data used to construct the curves in Fig 12 and 13 were obtained from tests of unblended coals, our studies have indicated that the values of blends of two or more coals in most cases lie within the looped curve of Fig 12. If further data confirm this hypothesis it will then be possible to select two coals which when blended together will yield a coke of the highest stability factor simply by obtaining the Gieseler maximum fluidity of each of the coals together with the rank of the coal from its analysis. With only a very small amount of data at hand there are further indications that the position of the point for the blend within the loop lies on or near a straight line connecting the values of the two coals blended. The fact that the location of the blend within the loop may not be proportionally distant from the two coals blended to the amount of each coal in the blend may possibly be explained by the fact that the loop in Fig 12 is the projection of the sloping curved plane that bounds the top of the curved plane shown in Fig 13. A large amount of work will be required to establish this concept.

BYPRODUCT YIELDS

For more than forty years Koppers Company, Inc. has been determining the byproduct yields of coals by the laboratory byproduct yield test. In Fig 14 the yields thus determined of gas, light oil, ammonium sulphate, and tar are shown in relation to coal rank. It will be noted that the maximum yields of each of these products occur approximately at the 40 pct volatile matter plus moisture on the dry, mineral-matter-free basis. The correlation of these values with coal rank is good and a fairly reliable prediction could be made from the knowledge of the coal rank.

Conclusions

In the foregoing study a procedure has been described by which the criteria used for the standard classification of coals can be arranged so that coals of all ranks lie on a single curved line. This development makes it possible to compare the coking behavior of one coal with another. In general, the coals of different ranks are lined up in accordance with their coking behavior. Exact correlation of coal rank with coking behavior has not been achieved, but the procedure described now makes it possible to determine the various causes for the lack of correlation. Such factors as ash content and differences in amount of petrographic constituents are suggested as causes. Not the least of the possible reasons for lack of correlation lies in the inaccuracies inherent in the methods of testing.

It is suggested in this study that the use of the classification charts together with data obtained from the Gieseler plastometer can be an important guide in the selection of coals. Although the selection of two or more coals for blending and the prediction of the characteristics of coke from the blend only have been indicated such predictions appear to be in the realm of possibility.

These studies have little more than described the framework of a simplified method for predicting the coking behavior of coals. That the successful culmination of this work is well justified can best be emphasized by the large amounts of money that are expended every year in testing the coking properties of coals. The correlations seem to substantiate the premise stated that the difference between coking properties of coals bears some quantitative relation to the analyses of the coals.

Acknowledgment

The authors are indebted to Mr. Fred Denig, Vice President and Director of Research, Koppers Company, Inc., for his continued interest and helpful suggestions and for permission to publish the information presented. We also wish to thank Messrs. J. F. Farnsworth, J. A. Hahn, J. K. McHugh, and H. B. Smith, Jr. for their assistance in obtaining data and preparation of the figures.

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An Evaluation of the Performance of Thirty-three Residential Stoker Coals

By JAMES B. PURDY* and HARLAN W. NELSON*

The great majority of stokers used in residential heating installations are of the clinkering type. Because of inherent characteristics of the underfeed combustion process as it occurs in these small stokers, and because of the wide range of operating conditions encountered in residential heating schedules, not all bituminous coals can be used successfully in residential underfeed units. The problem of providing consumer satisfaction has been attacked by the joint efforts of the stoker manufacturers and the coal industry; the stoker manufacturers have been concerned with improving their product to permit the use of a wider range of coals, and coal operators have made every effort to market selected and better prepared coals. In view of the wide range of physical characteristics and chemical properties of bituminous coals, the problems are real and are receiving increased attention.

Some twelve years ago a laboratory method was developed at Battelle Memorial Institute for the relative evaluation of coals for use in residential underfeed stokers. The method was developed in conjunction with work being done at that time for Bituminous Coal Research, Inc., and for individual coal producers, to obtain information on the characteristics of coal that determined the successful performance of the small stoker. Details of the equipment and test procedure were given by R. A. Sherman[1] in a paper presented at the University of Illinois Fifth Short Course in Coal Utilization.

During the past several years, the evaluation of over forty different coals has been determined by investigations sponsored at Battelle by coal operators in the eastern producing districts.

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[1] References are at the end of the paper.

The purpose of this paper is to compare the results and conclusions derived from the various tests with marketing experiences of the operators, and to discuss briefly the relationship of test results to standard laboratory determinations.

Equipment Used in Tests

Fig 1 shows schematically the equipment used in the evaluation tests. The stoker is a commercial model of conventional design, having a nominal feed rate of 30 lb per hr. A special laboratory furnace is used, which is tightly sealed to prevent the entrance of secondary air. The complete stoker-furnace assembly is mounted on platform scales which permits the measurement of combustible weight losses occurring during the test. The furnace stack, in which are a thermocouple, flue-gas sampling tube, and smoke meter, is connected to the laboratory exhaust breeching through a frictionless connection.

Through use of an auxiliary blower system, provision is made for the measurement of the air delivered by the stoker fan to the windbox. A water seal in the duct work eliminates inter-

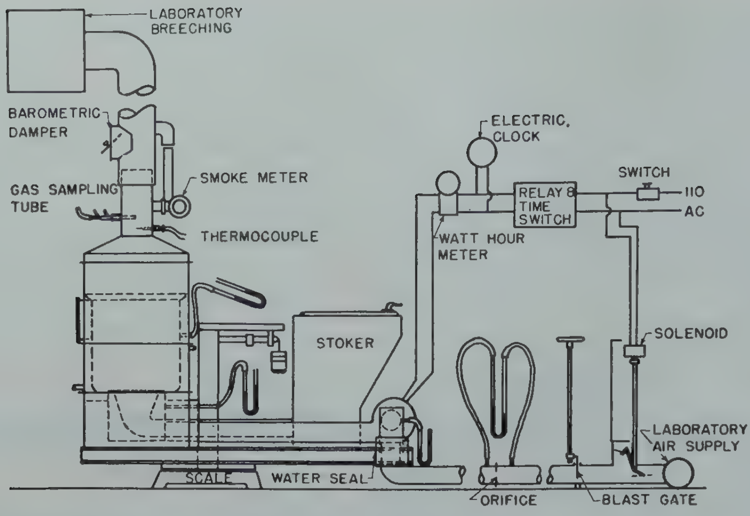

FIG 1—Diagram of test equipment.

ference with the process of weighing the stoker-furnace unit. A barometric damper is located in the furnace stack to assist in maintaining a constant draft. The damper is adjusted to give a draft of 0.05 in. of water during "off" periods of the stoker.

Smoke concentrations are measured by means of a smoke-density meter. The meter consists of a light source and photoelectric cell between which a sample of the flue gases is drawn at a constant rate. Variations in the absorption of light by smoke varies the output of the photoelectric cell, and the voltage is recorded continuously on a recording potentiometer.

Appropriate instruments are provided for control of hold-fire operation, for recording stoker running time and power consumption, and for measurement of overfire and windbox pressures. Flue-gas analyses are made at appropriate intervals throughout the test.

Test Procedure

At the start of the test, the hopper is filled with the coal to be tested, and the stoker is run until the retort is filled and coal is level with the top of the tuyeres. The feed of coal is then stopped, and 1 lb of charcoal is placed on top of the coal. Just prior to lighting the fire, one-half pint of kerosene is poured over the charcoal. The charcoal is then ignited and for a period of 5 min no coal is fed, but the stoker fan is permitted to run. At the end of 5 min the coal feed is turned on and allowed to run for the balance of the hour.

The air control on the stoker is set to furnish air at a rate 30 pct in excess of that theoretically required by the rate at which the coal under test is being fed. Any correction necessary to obtain this rate of air delivery is made during the first hour by means of the stoker air control. No further adjustment of the air is made throughout the test.

The schedule of operation for an evaluation test is designed to simulate the operation of a stoker under conditions of varying heat demand, as would be expected during a typical heating season. The test is normally of five days' duration. During the first and second days, operation consists of alternate periods of continuous and hold-fire operation; alternate periods of intermittent and hold-fire operation occupy the third day, and the fourth day consists wholly of hold-fire operation. Intermittent operation is

Table 1 . . . Analyses of Coals Tested

Test Number	Bed	State	Nominal Size, In.	Proximate Analysis, Per Cent					Sulphur, Per Cent	Calorific Value, Btu per Lb	Ash-softening Temp., °F	Free-swelling Index, Number
				Moisture	Volatile Matter	Fixed Carbon	Ash	Total				
1	Wallins	Ky.	1¼ × 0	2.5	40.1	51.8	5.6	100.0	0.8	13,630	2,560	
2	Wallins	Ky.	1¼ × ¼	2.8	40.0	52.4	4.8	100.0	0.8	13,700	2,250	
3	Upper Freeport	Pa.	¾ × ¼	1.1	33.8	56.0	9.1	100.0	1.6	13,800	2,360	
4	Pittsburgh	W. Va.	1 × 10 mesh	3.0	37.5	53.3	6.2	100.0		13,380	2,380	4.5
5	No. 2 gas	W. Va.	1 × 10 mesh	0.9	34.6	61.2	3.3	100.0	0.8	14,760	2,290	
6	Taggart	Va.	¾ × 0				3.1				2,210	8.5
7	Taggart	Va.	¾ × 0				4.5				2,310	8.5
8	Chilton	W. Va.	¾ × ⅜	1.1	31.9	58.5	8.5	100.0			2,700	7.0
9	Sewell	W. Va.	¾ × ¼	0.2	23.2	72.6	4.0	100.0		15,140	2,440	
10	Pittsburgh	W. Va.	¾ × ⅜	2.7	38.1	54.4	4.8	100.0		14,050		8.0
11	Elkhorn No. 3	Ky.	1 × ⅛	1.6	37.2	54.6	6.6	100.0		13,920		6.0
12	Elkhorn	Ky.	1 × ⅜	0.9	35.4	60.5	3.2	100.0			2,630	6.0
13	Elkhorn	Ky.	⅜ × ⅛	1.6	32.9	59.8	5.7	100.0			2,660	5.5
14	Elkhorn	Ky.	1 × ⅛	1.4	34.8	60.0	3.8	100.0			2,530	6.0
15	Elkhorn	Ky.	1 × 50 mesh	1.3	34.1	59.5	5.1	100.0			2,670	6.0
16	Mason	Ky.	1 × ¼	0.6	37.8	56.4	5.2	100.0	1.6	14,160	2,420	5.0
17	Mason	Ky.	1 × ¼	1.7	36.3	58.3	3.7	100.0		14,089	2,530	4.0
18	No. 11	Ky.	¾ × 48 mesh				6.1				2,200	4.0
19	High Splint	Ky.	1 × 5/16	0.7	39.0	52.5	7.8	100.0	1.0	13,400	2,730	3.5
20	Blue Gem	Ky.	1¼ × ⅜	1.9			6.0		1.6	13,680		3.5
21	No. 2 gas	W. Va.	1 × ⅜	1.7	35.7	55.7	6.9	100.0	1.4	13,951	2,680	8.5
22	(Unknown)	Ind.	¾ × ⅜	7.5	41.0	44.0	7.5	100.0	3.0	12,397	2,190	
23	Hazard No. 4	Ky.	1 × ⅜	1.9	37.6	55.2	5.3	100.0	0.8	13,961	+2,700	5.0
24	Hazard No. 4	Ky.	1 × ⅜	2.2	37.5	55.2	5.1	100.0	0.9	13,940	2,410	4.5
25	Elkhorn No. 4	Ky.	1 × ⅜	1.1	38.5	55.6	4.8	100.0	1.0	13,970	2,500	5.0
26	Hernshaw	W. Va.	1 × ⅜	0.8	32.3	63.6	3.3	100.0	0.7	14,690	2,750	7.5
27	Rich Mountain	Tenn.	1¼ × 3/16	1.3	35.7	56.1	6.9	100.0	1.0	13,730	+2,800	4.5
28	Shannon	Va.	1¼ × ¼	1.9	33.6	53.4	11.1	100.0	1.1	13,283	2,280	6.5
29	No. 2	Ky.	1 × ⅜	1.3	37.3	46.5	14.9	100.0	5.1	12,470	2,310	3.5
30	No. 7	Ky.	1 × ¼	6.0	38.9	51.4	4.6	100.0	1.8	12,780	2,190	4.0
31	Powellton	W. Va.	1 × ⅜	1.8	35.5	57.1	5.6	100.0	1.1	14,330	2,420	8.5
32	Hazard No. 4	Ky.	1 × ¼	1.2	38.0	55.2	4.7	100.0	2.6	14,130	2,340	5.0
33	Clintwood	Ky.	1 × 4 mesh	0.4	29.1	63.9	6.6	100.0	1.1	14,360	2,700	8.0

scheduled for the whole of the fifth day.

For the parts of the schedule indicating continuous operation, the stoker is started and stopped manually. For hold-fire operation, the stoker is started once each hour by a timer control and is stopped by a potentiometer controller when the temperature of the flue gases has reached 20 mv, or approximately 900°F. During intermittent operation, the stoker is manually started at intervals of 20 min, the duration of each period of operation being determined by the time required to burn one-third of the quantity of coal necessary to release 187,500 Btu per hour. The stoker is stopped when the required weight loss is attained, and remains idle until the start of the next 20-min period.

As a test is always started with a completely clean hearth, the fuel bed is not disturbed until the evening of the fourth day, when a probe is made with clinker tongs to remove any clinker from the fuel bed. At the conclusion of the test the fire is quenched with water and the fuel bed is removed from the furnace. Pieces of clinker larger than 3 in. in size, and all coke retained on a 1-in. round-hole screen, are separated from the fuel bed and weighed.

Bases for Evaluation

In evaluating a stoker coal certain tangible factors must be selected on which a judgment of the coal may be based. These factors are chosen because of their importance in determining the degree of satisfaction that the consumer will obtain in using the coal, from the standpoint of manual attention, uniformity of heating, comfort, cleanliness, and cost. The main factors used as a basis for evaluation in this test are:

1. Initial ignition.
2. Clinker formation.
3. Coke formation.
4. Rate of fuel consumption during hold-fire operation.
5. Resistance of fuel bed to the flow of air.
6. Uniformity of burning during intermittent operation.
7. Smoke concentration.

Coals Tested

Over 40 tests have been conducted, and their results evaluated by the method described. A few of these were short-term tests consisting of only the first and last days of operation. Of the 5-day tests, the results of 33 are included in the present study. With one exception, the coals were all high volatile, and were largely from the eastern producing districts.

Table 1 presents the laboratory-determined characteristics of the coals used in these tests. In addition to the proximate analysis, the sulphur, calorific value, ash-softening temperature, and free-swelling index are given. The nominal size of the coal is indicated.

Results of Tests

At the conclusion of each test, average values are determined for components of the flue-gas analysis, pressure measurements, smoke concentrations during "off" periods, and length of hold-fire running periods. These and other directly determined values are then subjected to analysis.

Table 2 lists values of many of these data for the coals tested. When reporting the results of tests, in addition to listing data shown in the table, plots of some groups of data are made for purposes of showing their degree of variability during the course of the test. Wide variations in the length of operating times during intermittent operation, for example, are indicative of excessive coke-tree formation, dirty fuel bed, or general lack of response to heat-demand operation, but average values alone do not indicate this variability. Supplementary data and information, including photographs and visual observation of the fuel bed, strength of the clinker formed, and the height and type of coke trees, are not subject to numerical evaluation but are useful in making an overall interpretation of results.

The ratings of initial ignition shown in Table 2 represent approximate evaluations of the curve of flue-gas temperatures recorded during the first hour of the test. The word descriptions shown furnish an initial indication of the ignitibility and coking characteristics of the coal under test. The curves for initial ignition are discussed more fully in a later section of the paper.

For the coals tested, the range of values for the drop in pressure through the fuel bed was from 0.52 to 1.54 in. of water. The values are indicative of the general condition of the fuel bed, and any coal that has a fuel bed resistance greater than 1.5 in. of water is considered to give difficulty in burning. Of the coals tested, only one showed values above this maximum.

As previously stated, during hold-fire operation, the stoker is started once each hour and allowed to run until a stack temperature of 900°F is reached. Under this procedure, the

Table 2 . . . Summary of Results from Evaluation Tests

Test Number	Bed	State	Initial Ignition	Pressure Drop through Fuel Bed, In. H_2O	Hold-fire Time, Min.	Smoke Concentration, Average	Coke in Fuel Bed, End of Test, Lb	Clinker Removed, Per Cent of Ash Released
1	Wallins	Ky.		1.16		11.8	4.1	16.8
2	Wallins	Ky.		0.86		13.6	2.6	27.2
3	Upper Freeport	Pa.		1.54	4.5	27.6	14.5	24.0
4	Pittsburgh	W. Va.		1.08	5.4	21.6	9.6	47.6
5	No. 2 gas	W. Va.		0.84	4.1	15.6	16.5	27.9
6	Taggart	Va.	Satisfactory	0.76	4.3	12.0	4.3	13.4
7	Taggart	Va.	Very satisfactory	0.91	5.3	16.7	9.6	21.0
8	Chilton	W. Va.	Satisfactory	1.26	6.8	14.0	3.8	14.0
9	Sewell	W. Va.	Poor	1.19	4.2	7.5	16.1	25.3
10	Pittsburgh	W. Va.	Poor	0.79	5.4	33.4	4.9	16.3
11	Elkhorn No. 3	Ky.	Very satisfactory	1.01	5.8	14.7	10.8	28.4
12	Elkhorn	Ky.	Very satisfactory	0.52	4.6	14.7	8.4	12.5
13	Elkhorn	Ky.	Satisfactory	0.85	4.3	9.4	16.4	39.8
14	Elkhorn	Ky.	Satisfactory	0.65	4.2	10.9	7.8	26.2
15	Elkhorn	Ky.	Satisfactory	0.86	4.3	14.4	19.5	36.6
16	Mason	Ky.	Poor	1.05	5.3	15.8	9.1	37.1
17	Mason	Ky.	Very satisfactory	0.72	4.1	10.6	4.1	21.3
18	No. 11	Ky.	Satisfactory	0.97	4.3	17.9	8.4	31.0
19	High Splint	Ky.	Very satisfactory	0.77	4.8	5.0	3.6	17.8
20	Blue Gem	Ky.	Very satisfactory	0.77	4.9	12.8	13.7	53.3
21	No. 2 gas	W. Va.	Poor	0.81	6.0	19.0	19.0	16.2
22	(Unknown)	Ind.	Satisfactory	0.68	6.3	16.0	6.6	21.2
23	Hazard No. 4	Ky.	Satisfactory	1.09	5.4	12.0	7.2	23.5
24	Hazard No. 4	Ky.	Very satisfactory	0.97	4.5	16.0	7.4	14.9
25	Elkhorn No. 4	Ky.	Satisfactory	0.82	5.7	20.0	8.6	14.0
26	Hernshaw	W. Va.	Very satisfactory	0.78	4.6	11.0	5.4	11.6
27	Rich Mountain	Tenn.	Poor	1.26	6.3	21.0	11.9	17.5
28	Shannon	Va.	Poor	0.90	5.9	15.0	7.5	26.0
29	No. 2	Ky.	Poor	1.28	7.1	17.0	42.8	45.4
30	No. 7	Ky.	Very satisfactory	0.63	5.1	15.0	3.9	18.4
31	Powellton	W. Va.	Poor	0.93	5.7	22.0	17.5	36.0
32	Hazard No. 4	Ky.	Satisfactory	0.63	5.2	20.7	10.2	28.6
33	Clintwood	Ky.	Poor	1.04	6.2	18.6	15.5	30.5

time of operation is governed by characteristics of the coal. A cut-off temperature of 900°F may be too high, as many coals will be capable of holding the fire at a lower temperature; however, time would not permit trial of minimum settings, and the results may be compared on a relative basis. For purposes of evaluation, coals having time values below 5 min per hr are considered satisfactory. The range of values shown in Table 2 was from 4.1 to 7.1 min per hr. Values of less than 5.0 min were determined for 14 of the coals tested, and 6 of the coals required operating periods averaging over 6.0 min.

The values of smoke concentration were determined as the average density of smoke during the "off" periods of intermittent operation. Only coals showing average values above 20 on the logarithmic scale of densities would be apt to produce troublesome smoke during "off" periods, as combustion of volatile gases and dilution of smoke by air admitted over the fire would substantially reduce the smoke concentrations in a residential heating installation.

Values of the weight of coke remaining in the fuel bed at the end of a test give a numerical indication of the extent of coke formation and the depth of the fuel bed. Comparisons are significant only when all tests are run with closely similar percentages of excess air. The range of values found during the present tests was from 2.6 to 42.8 lb; 8 of the coals produced 15.0 lb of coke or more.

The measure of clinker formation represented by data listed in Table 2 is the percentage of the total ash released during the test, which is formed into clinker. Percentage values of 20 pct or more are considered to indicate satisfactory clinker formation.

Table 3 . . . Summary Information Regarding Conclusions from Evaluation Tests and Marketing Experience of Coal Operators

Test Number	Bed	State	Classification of Laboratory Test[a]	Reason for Classification	Experience in Marketing by Sponsor Operator
1	Wallins	Ky.	S		No information
2	Wallins	Ky.	S		No information
3	Upper Freeport	Pa.	U	Strong coke-tree formation	Does not market residential stoker coal
4	Pittsburgh	W. Va.	S		No information
5	No. 2 gas	W. Va.	S		Marketed successfully
6	Taggart	Va.	S		Marketed successfully
7	Taggart	Va.	S		Marketed successfully
8	Chilton	W. Va.	U	Poor clinker formation and hold-fire characteristics	Not marketed for use in residential stokers
9	Sewell	W. Va.	S		No information
10	Pittsburgh	W. Va.	S−	Sluggish ignition, length of hold-fire periods above average, low clinker formation	No information
11	Elkhorn No. 3	Ky.	S		Marketed successfully
12	Elkhorn	Ky.	S−	Poor clinker formation	No information
13	Elkhorn	Ky.	S		No information
14	Elkhorn	Ky.	S		No information
15	Elkhorn	Ky.	S		No information
16	Mason	Ky.	S		Marketed successfully, principally in northern markets
17	Mason	Ky.	S−	Relatively poor clinker formation	Facilities not yet available for preparation of stoker coal
18	No. 11	Ky.	S		Marketed successfully
19	High Splint	Ky.	S−	Clinker formation questionable	Marketed successfully
20	Blue Gem	Ky.	S		Mine now closed, plan to prepare stoker coal when re-opened
21	No. 2 gas	W. Va.	S−	Sluggish ignition, strong coke formations, clinker formation low	No information
22	(Unknown)	Ind.	S		New mine, upon completion of tipple will market stoker coal
23	Hazard No. 4	Ky.	S		Small mine, output not yet justifies preparation of stoker sizes
24	Hazard No. 4	Ky.	S−	Inadequate clinker formation	Has been marketed successfully
25	Elkhorn No. 4	Ky.	S−	Inadequate clinker formation	No stoker coal prepared for market
26	Hernshaw	Ky.	S−	Poor clinker formation	Preparation facilities lacking, no stoker sizes prepared
27	Rich Mountain	Tenn.	U	Sluggish ignition and hold-fire characteristics, high in coke formation, low in clinker formation	Only run-of-mine being marketed
28	Shannon	Va.	U	Sluggish ignition and hold-fire characteristics, high ash content	No stoker sizes prepared
29	No. 2	Ky.	U	Poor ignition and hold-fire characteristics, sluggish fuel bed, high ash content, heavy coke formation	Decision made not to prepare stoker coal
30	No. 7	Ky.	S		Stoker sizes not being prepared at present time
31	Powellton	W. Va.	S		Stoker sizes not being prepared at present time
32	Hazard No. 4	Ky.	S		New tipple under construction, stoker sizes will be prepared
33	Clintwood	Ky.	S		Equipment for preparation of stoker sizes not yet installed

[a] S, Satisfactory. S−, Not completely satisfactory. U, Unsatisfactory.

Relation of Conclusions Drawn from Test Results to the Marketing Experience of Operators

From consideration of the test results, conclusions are drawn with regard to the degree of satisfaction to be expected if the coal under test were to be marketed. Many of the coals tested could immediately be classed as satisfactory or unsatisfactory beyond reasonable doubt; others were deficient in performance by one or more criteria, and were given an intermediate or limited classification.

Table 3 lists the coals tested, their classification based on test results, the principal reasons for the classification, if unsatisfactory or not completely satisfactory, and the marketing experience of the coal operators sponsoring the tests. Of the 33 coals tested, 20 were determined to be satisfactory for use in residential stokers, 5 were listed as being unsatisfactory, and 8 were given limited ratings.

The chief cause for limited ratings was inadequate clinker formation. Although satisfactory in other respects, use of coals in the restricted classification might be expected to result in complaints from consumers because of the accumulation of unfused ash in the furnace. Such coals might be marketed successfully in the northern regions, but might prove unsatisfactory in regions where climatic conditions require frequent and prolonged operation at low or hold-fire rates. Other factors adversely affecting the evaluation of the coals were excessive coke-tree formation, sluggish ignition and response during intermittent operation, high ash content, and high average smoke concentrations during "off" periods of stoker operation.

As shown in Table 3, there was gen-

eral agreement between conclusions drawn from results of evaluation tests and the marketing experiences of the coal operators. Those coals classed as unsatisfactory have not been prepared by the respective operators in residential stoker sizes. Most of the coals rated as being satisfactory are either being marketed successfully, or will be marketed when preparation facilities become available. Of those rated as not being completely satisfactory, three have been marketed successfully; the others are not being prepared now in residential stoker sizes, or information regarding them is lacking.

The agreement between results of the evaluation tests and marketing information is reasonably good and appears to justify use of the method. Because it is a relatively short-time test, an appraisal of a coal can be accomplished much more quickly than if adequate field tests were attempted. Many of the requests for tests come in summer months when operators are interested in the market for the following heating season.

The method possesses limitations, particularly in regard to the evaluation of clinker formation with coals of intermediate ash-fusion characteristics. With coals of this class having low ash content, the fraction of the ash formed into clinker may be unduly low, although if the test were continued, the fraction would become greater. Admittedly, judgment of clinker formation is less susceptible to evaluation than any other factor involved in the test.

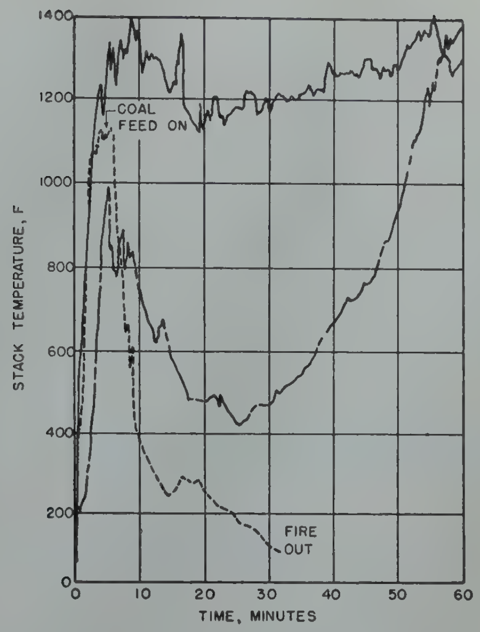

FIG 2—Flue-gas temperature records, initial ignition period.

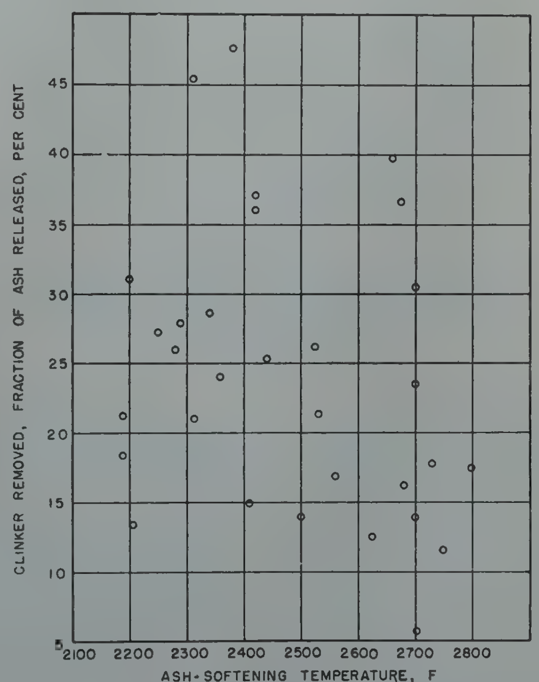

FIG 3—Relation between clinker formation and ash-softening temperature.

Relation of Results of Test to Standard Laboratory Determinations

Analysis of the data from the evaluation tests has furnished an opportunity for attempts to relate these results to data determined for the coals by standard laboratory determinations. Studies of this kind have been made by Shotts,[2] on Alabama coals, and by Helfinstine and Boley,[3] on Illinois coals. There is a natural interest in correlations of this kind, because of the possibility for simplification of an evaluation test procedure.

INITIAL IGNITION

The variation in the time required for the flue-gas temperature to reach an equilibrium value, as shown by the record of temperature during the first hour of the test, is considered an indication of the performance of the coal with respect to ignitibility and coking characteristics of the coal. Although not a measure of individual characteristics, it gives a preliminary indication of difficulties to be expected in maintaining ignition during operation under hold-fire conditions.

Fig 2 presents three examples of initial ignition curves from among the coals tested. These curves emphasize the difference that can exist between bituminous coals in reaching an equilibrium flue-gas temperature value during the initial ignition period. The top curve represents a coal having very good ignition characteristics; the maximum flue-gas temperature was reached early in the hour and

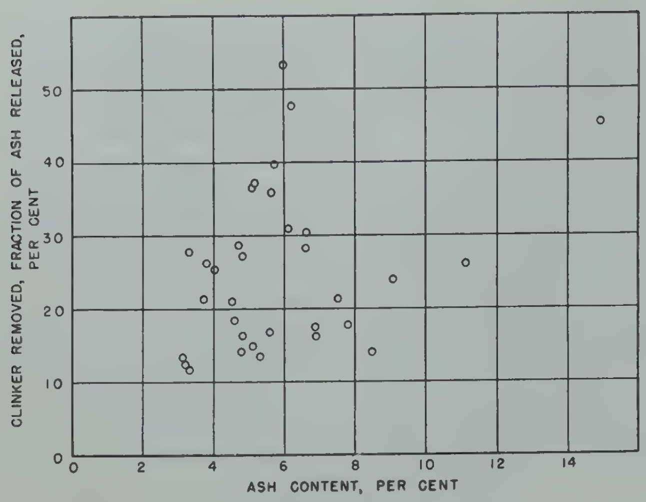

FIG 4—Relation of ash content to clinker formation.

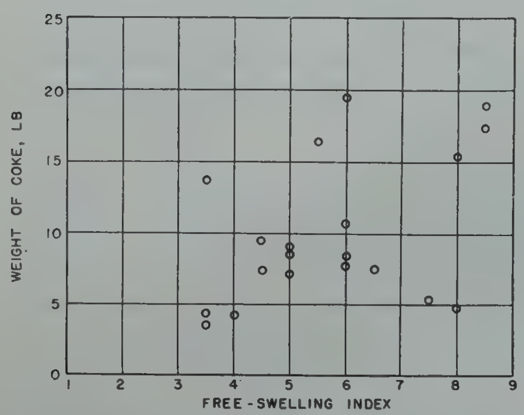

FIG 5—Relation between coke accumulation and the free-swelling index.

declined only slightly during the balance of the hour. The middle curve shown as a broken line, represents a coal having sluggish ignition characteristics; the flue-gas temperature declined sharply during the first 25 min and did not recover to its maximum value until the end of the first hour. The bottom curve represents unsatisfactory ignition in that the fire had gone completely out at the end of 30 min of operation.

An attempt was made to relate data from curves of initial ignition to free-swelling indexes and to the time required for operation of the stoker during periods of hold-fire. This was done by plotting the variance of the flue-gas temperature during the initial ignition period against the free-swelling index, and against the average running time during hold-fire operation. No direct relationship was found to exist between these data; hence, the temperature variation during ignition cannot be used alone as a criterion of coking or hold-fire characteristics.

Some years ago, an investigation of ignitibility of bituminous coals was made at Battelle. In that study, ease of ignition in residential stokers was compared with ignition temperatures determined by the method developed by The Coal Research Laboratory at the Carnegie Institute of Technology. The correlation between these properties was not good. It is concluded that ease of ignition depends not only on ignition temperature, but also on other factors, chiefly coking characteristics and size variation.

CLINKER FORMATION

Results of clinker formation in the evaluation test are expressed as the percentage by weight of the total ash released during the test which is formed into plus 3 in. clinker. If values for these percentages are 20 pct or higher, the coals may be expected to give satisfactory service to the consumer in this respect; below this value, the clinkering characteristics of the coal are open to question.

Comparison of clinker values obtained in these tests and ash-softening temperature shows no correlation. Fig 3 presents a graph of the relation between ash-softening temperature, as determined by the standard laboratory method, and the clinker removed from the fuel bed in the burning tests. Poor correlation is evident between these data. Similar results have been obtained by Helfinstine and Boley.[3]

The ash content of the coal as determined by the proximate analysis definitely predicts the degree of manual attention that will be required when burning the coal, due to the removal of clinker and unfused ash.

Fig 4 presents the relation between the ash content of the coal and the clinker formation. The relationship here is little better than that shown in Fig 3; however, no single laboratory-determined characteristic can accurately predict the performance of a stoker coal with respect to clinker formation.

COKE FORMATION

In the evaluation test, a numerical measure of coke formation is obtained by the weight of plus 1 in. coke remaining in the fuel bed at the end of the test. The standard laboratory test used to obtain an indication of the coking tendencies of a coal is the free-swelling test as prescribed by ASTM designation D-720-46.

Fig 5 shows the relation between these two methods. The correlation is not as close as might be expected, though the results are in agreement with those of Shotts.[2] The reason for this lack of correlation is largely the result of variations in the reactivity of the coke formed.

RATE OF FUEL CONSUMPTION DURING HOLD-FIRE OPERATION

The stoker running time necessary to maintain the fuel bed in good condition is of importance to the consumer for reasons of economy and comfort, as it determines the amount of coal that will be burned during periods when there is no demand for heat.

An investigation was made of the relation between the time required to hold fire and laboratory-determined characteristics. Between time required to hold fire and the free-swelling index no relationship could be determined. A

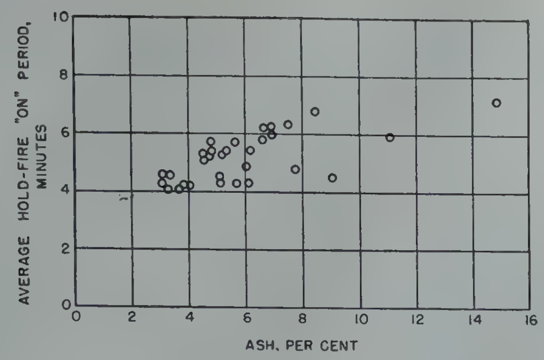

FIG 6—Relation between ash content and average time "on" during hold-fire operation.

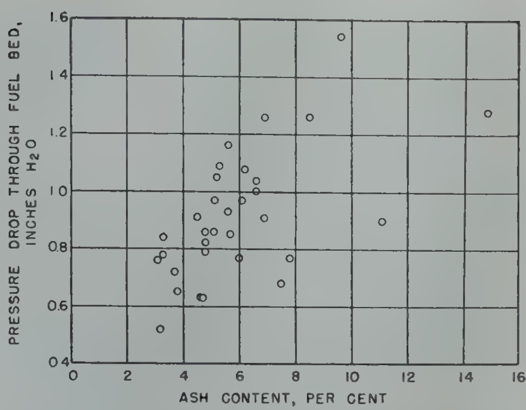

FIG 7—Relation between percentage drop through fuel bed and ash content, per cent.

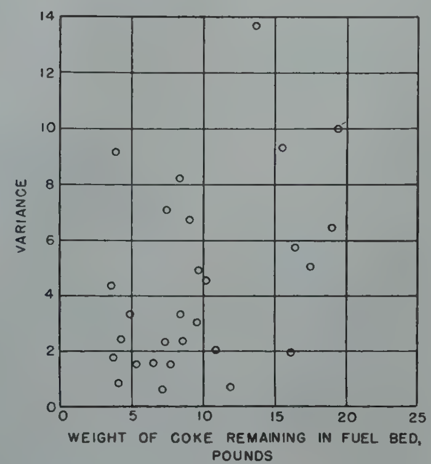

FIG 8—Relation between variance in time "on" during intermittent operation and weight of accumulated coke.

plot of the relation between hold-fire time and the ash content is shown in Fig 6. These data indicate that, in general, the hold-fire time is increased slightly with increase in the ash content of the coal.

RESISTANCE OF THE FUEL BED TO THE FLOW OF AIR

The resistance of the fuel bed to the flow of air is of interest, as it may reflect the general condition of the fuel bed and the ability of the stoker fan to furnish the necessary volume of air. The fan capacity of residential stokers is sufficient to overcome maximum fuel bed resistances of 1.50 to 1.75 in. of water. With suitable automatic air controls, the delivery of air will be maintained up to this pressure. The highest resistance found during the present study was 1.54, and, for most of the coals tested, values were well below the limiting pressure.

The principal factors contributing to fuel bed resistance are coking properties, size of the coal, and ash content. No correlation was found between free-swelling index, as indicative of coking properties and fuel bed resistance. Fig 7 presents a plot showing the relation of fuel bed resistance to the ash content of the coals. The effect of increasing ash content upon the resistance offered by the fuel bed is evident in this graph.

UNIFORMITY OF BURNING

The ability of a stoker coal to respond to abrupt demands for heat, and to burn uniformly during periods of heat demand are very important factors in providing satisfactory operation for the consumer. The uniformity of burning is judged mainly by the performance of the coal during the period of intermittent operation on the last day of the test. A measure of the performance during this period is obtained by the operating time necessary to liberate a given quantity of heat, and the uniformity of these values for successive operating intervals.

Fig 8 shows the relation between the variance in time "on" during intermittent operation and the weight of coke remaining in the fuel bed at the end of the test. The trend of the data in this graph shows that increased quantities of coke are reflected in increased variations in the uniformity of burning. Fig 9 presents a similar plot for clinker formation and variance. Here it is shown that the amount of clinker formed had much the same effect on the uniformity of burning.

A study of data relating the free-swelling index and the average "on" period during intermittent operation showed a complete lack of correlation.

SMOKE CONCENTRATION

When the stoker is in operation during a test, under conditions of at least 30 pct excess air as normally employed, smoke concentrations are of no consequence, even when burning high-volatile coals. Smoke emission is noticeable only for a brief period after the stoker shuts off.

Of the laboratory-determined characteristics, volatile matter would appear to be the most closely related to smoke concentration; the volatile matter content usually is associated with the relative smokiness of a coal.

Fig 10 shows in graphic form the relation of volatile matter and smoke concentration for the coals tested. In general, the correlation of the data for these two characteristics is poor, indicating that volatile matter alone cannot determine the amount of smoke that will result from burning a coal in residential stokers.

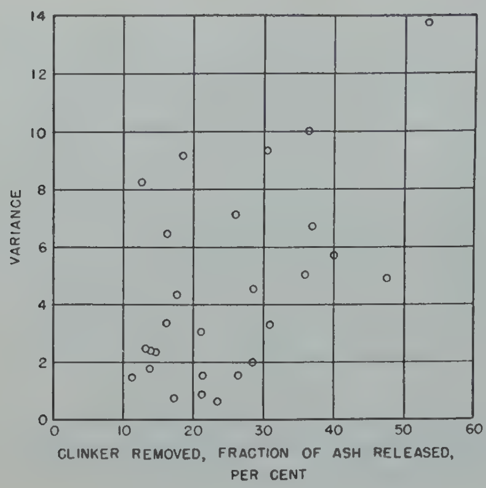

FIG 9—Relation between clinker formation and the variance.

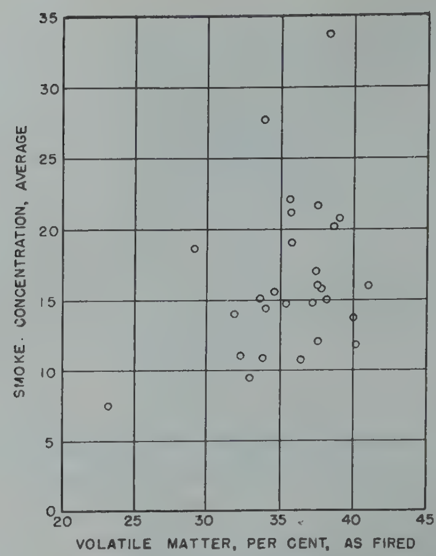

FIG 10—Relation between smoke concentration and volatile matter content.

Conclusions

A study of data obtained during laboratory tests to determine the suitability of bituminous coals for use in residential underfeed stokers of the clinkering type has led to the following general conclusions:

1. Performance data obtained during the test furnish useful and practical information upon which to judge the performance of stoker coals.

2. Conclusions derived from the results of evaluation tests are in general agreement with use experience of the coals in the field. The method is of practical use in this regard.

3. Few direct relationships were found between data furnished by the evaluation test and determinations by standard laboratory tests. This is particularly true with regard to coke formation and clinker formation, two of the most important factors in determining the degree of satisfaction to be obtained in applications of the small underfeed stoker.

It has not been the intention in this paper to prescribe this evaluation test as the best method for the evaluation of coals for use in residential stokers. That there are other test methods having similar objectives serves to point out the necessity for full-scale laboratory tests.

At the present time, a standard test procedure for the evaluation of coals for use in residential stokers is not available. For the past several years a joint committee, appointed from the membership of the Residential Stoker Committee of Bituminous Coal Research, Inc., and from the Engineering and Research Committee of The Stoker Manufacturers Association, has been working to develop a standard procedure for testing and evaluating bituminous stoker coals. This procedure has been completed to the extent that it has been drawn up in tentative form. It is now planned to install the equipment in several laboratories and to run check tests, using portions of identical lots of coal, to determine the degree of reproducibility offered by the method. Completion of this standard test procedure should greatly facilitate the evaluation of coals for use in residential stokers.

Acknowledgments

The cooperation of coal operators sponsoring the coal evaluation tests, in granting permission to use data and results from their tests, is gratefully acknowledged. Acknowledgment is also made to Carroll F. Hardy, Chief Engineer, Appalachian Coals, Inc., for his interest and timely suggestions on the preparation of this paper. Special thanks are due to Ralph Sherman, Assistant Director, Battelle Memorial Institute, who was largely responsible for development of the evaluation procedure, for helpful suggestions and advice.

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DISCUSSION

E. R. KAISER*—As a former fuel engineer assigned to residential stokers at Battelle, the writer has read with considerable interest the paper by Messrs. Purdy and Nelson. It is gratifying to note that earlier test techniques have been developed and that criteria found important in the years 1935 to 1938 are still important in judging the performance of stoker coals for residential heating.

The authors have plotted a number of the primary coal data singly against performance data of the stoker in an effort to establish relationships. Unfortunately, wide scattering of points usually resulted, which did not illustrate what experience generally indicates to be the case. For example, Fig 3 indicates almost no trend in the percentage of released ash converted to clinker with change in ash-softening temperature. Other factors, such as fuel bed temperatures and zones of heat release must have influenced the results.

In the present state of our knowledge, we cannot explain all of the reasons why suitable stoker coals perform satisfactorily, nor why one satisfactory coal may be better than another. It would therefore

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be better to approach the subject from the side of unsatisfactory coals whose properties might be more marked and therefore easier to evaluate. On examining the data for sixteen unsatisfactory coals tested at Battelle, not all of which were reported in the subject paper, it was concluded with one of the authors as follows: Unsatisfactory coals for residential stokers have one or more of the following general characteristics in the Battelle test:

1. Slow ignition with less than 20 lb of coal burned during the first hour.
2. Over 8 pct ash content.
3. Over 2600°F ash-softening temperature.
4. Less than 20 pct of the ash released converted to clinker.
5. A smoke reading of over 12 units average during the "off" period.
6. A windbox pressure exceeding 1.25 in. w.g.
7. Weight of coke at end of test exceeding 10 lb.
8. Free-swelling index exceeding 8.0.
9. It is assumed that the top size does not exceed 1¼ in. and that the amount of minus ¼ in. coal does not exceed that of normal resultant coals.

Returning to the question of correlating stoker performance with basic laboratory tests, the stoker performance is so complicated that it is not reasonable to expect the performance factors to vary with single coal factors. At least two or three coal characteristics are at work at the same time in affecting clinker formation. These factors are discussed under the same headings as used by the authors.

Ignition Properties

A coal with poor initial ignition shows symptoms of poor ignition later during "on" periods and hold-fire. Coking characteristics and size consist may be factors, but something inherent in the coal other than that measured by the Coal Research Laboratory method seems to be present. Perhaps it is liberation of CO_2 and water vapor from the coal, with absorption of heat, when the coal is subjected to ignition. Weakly coking coals from the Indiana-West Kentucky field sometimes exhibit poor ignition. Outcrop coals are notorious in this respect. Additional basic research is needed on this subject.

Clinker Formation

Clinkering of coal ash is evidently affected not only by the ash-softening temperatures but by the temperatures in the ash zone. A hot fire in the region beginning immediately above the tuyeres and extending at least 5 in. upward will promote clinkering. A coal that ignites slowly and produces coke rather than heat is not likely to promote clinkering. A high ash content does not help rapid ignition nor rapid combustion. Furthermore, the ash-softening temperature has the limitation of not representing the ash in its segregated or heterogeneous state on being released from the coal.

Coke Formation

The poor correlation between free-swelling index and weight of coke in the fuel bed is probably caused by the additional factors of slow ignition in the retort and reactivity of the coke formed.

Plasticity of the coal in the initial stage of coking is probably the dominant factor, with ignition second, and reactivity of the coke third. The plastic property obviously must be first in importance in forming the coke tree. Slow ignition causes green and unoxidized coal to move up into the heat zone where the temperatures are better for coking than for rapid combustion. Once coke is formed, air cannot penetrate the formation.

Because of hold-fire conditions and the cooling effect of the furnace walls, particularly in household furnaces, the coke loses ignition and often will not burn until it falls to the hearth and is reignited by the flames issuing from the annular space above the tuyeres.

Hold-fire Operation

The ignition property of the coal must again be a dominant factor in the length of time the stoker must be operated to brighten the fire. It is therefore understandable why the free-swelling index did not alone prove to be the factor. A small fuel bed of hot coal with a low fuel maintenance is all that is necessary. A heavy fuel bed of smoldering coal results if the stoker feed screw continues to deliver coal that will not ignite at the rate fed.

Fuel Bed Resistance to Air Flow

Coals that ignite slowly often cause high windbox pressures. Green coal floods the tuyeres rather than burning away promptly in the air jets. Low windbox pressures result when the combustion is rapid and a free annular space is left around the coke tree and inside the clinker ring.

Any unclinkered ash that falls into the annular space contributes to the air resistance. The windbox pressure also increases gradually with the amount of clinker in the bed simply by increasing the height of the annular space through which the gases must flow.

Uniformity of Burning

Coals that ignite promptly and do not coke too strongly burn with a small fuel bed. The uniformity of burning is good, neither too much nor too little, but approximately at the rate the coal is fed.

The combination of strong coking and slow ignition makes for large coke masses. While the coke is being formed, the rate of burning is slow. After the coke mass breaks down and is reignited, the burning is fast. The excess air varies inversely with the rate of burning. The extreme case occurs during the "on" period following the removal of clinker from a hot fuel bed. The coke is broken down in the process and reactive surfaces are exposed. Long flames and appreciable carbon monoxide in the flue gas result for a few minutes following the resumption of full rate combustion.

Smoke Concentration

Poor ignition can also contribute to off-period smoke. Smoke increases with the amount of green coal and partially devolatilized fuel above the retort, particularly at the center of the fuel mass. Prompt ignition and intense burning near the tuyeres reduce the coal left to cause smoke when the air flow is shut down for the "off" period.

Conclusion

The foregoing indicates strongly that the ignition property of bituminous coals should be investigated in a basic way to establish a laboratory method that would help not only in evaluating coals but in analyzing the performance of coals in combustion. Tests conducted at Battelle several years ago with fuel beds ignited from above and supplied with air from below showed promise of obtaining useful results.

Further study might well be given to the noncombustible gases liberated from coal on heating above room temperature. The heat required to raise the temperature of the coal, including that for liberating the gases, should also be measured. This heat is not important in relation to the total calorific value of the coal, but in underfeed ignition the amount of heat radiated in the ignition zone is small enough to make this factor highly important. If the tests could be conducted on the same coals used in the laboratory stoker tests it may be possible to develop a better correlation of all of the data obtained thus far, and to place a measured value on the ignition factor.

J. B. PURDY (*author's reply*)—Mr. Kaiser's knowledge of the background of these tests has provided a very interesting discussion. The limiting values generally found to be present among coals not considered suitable for use in residential underfeed stokers of the clinkering type, have been clearly summarized.

Of special interest are the comments, made in Mr. Kaiser's conclusion, regarding the need for additional basic research on the ignition properties of various coals. It is hoped that additional work can be accomplished along these lines in the near future.

Work of the U. S. Geological Survey on Coal and Coal Reserves

By PAUL AVERITT*

The U. S. Geological Survey has been actively engaged in work on coal for more than 50 years. During this long period we have released more than 300 publications containing information about coal and coal reserves, including details on the thickness, distribution, and quality of coal, and the structure and stratigraphy of the coal-bearing rocks. We have also cooperated with state surveys on many similar publications. For many areas these reports are the only reliable published data on our resources of coal.

During the last several years we have substantially increased our activities above the wartime low. We now have field work in progress, or reports in preparation, on 13 detailed mapping projects distributed as follows:

1. Pennsylvania anthracite field
2. Casselman Basin, Md.
3. Leslie County, Ky.
4. Deep River field, N. C.
5. Coosa field, Ala.
6. Powder River field, Mont.
7. Spotted Horse field, Wyo.
8. Yampa field, Colo.
9. Paonia field, Colo.
10. Trinidad field, Colo.
11. Durango field, Colo.
12. Chaco River area, N. M.
13. Lewis and Thurston County fields, Wash.

For two of these areas, the Paonia field, Colo., and the Coosa field, Ala., we have published recently preliminary reports, and final reports are in preparation. In addition, we are nearing completion of a coal reserve study of Montana, and are beginning similar studies in several other states, as part of a program of revising our estimate of national coal resources on the basis of presently available data.

Factors Involved in Making Estimates

Our responsibilities include detailed field mapping to secure information on areas of outcrop of coal beds, their range in thickness, nature of roof rock, amount of overburden, correlation of the beds, stratigraphy and structure of the coal-bearing rocks, and the preparation of coal reserve figures based on these data. The computation and estimation of resources of coal in the ground, like that of resources of other mineral commodities, involves consideration of many additional purely geological factors that can be appraised best through such regional studies. Regional variations in the thickness and texture of rocks overlying and underlying coal beds, for example, give evidence as to the probable position of the edges of the basin of coal deposition, or may aid in the delineation of sand-filled channels that sometimes cut out coal beds. It is apparent that the detailed field studies, in which data are gathered, are the basis for coal reserve studies, and in our present program these two phases of work are being carried on together.

Previous Coal Reserve Estimates

For many years we have been aware

White Sulphur Springs Meeting, November 1948.

Paper presented at Joint Fuels Conference of Coal Division AIME and Fuels Division ASME.

TP 2571 F. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before July 30, 1949. Manuscript received Dec. 7, 1948.

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* U. S. Geological Survey, Washington, D. C.

of the need for a substantial revision of the Campbell reserve figures, which were last revised by him in 1928, and which have been used widely since that time with only minor changes. We had little opportunity, or funds, for this work during the years following 1929. However, we did accumulate much new data, particularly on Montana, and when we were able to start a new reserve tabulation, we began our national program with a reappraisal of our accumulated data on reserves in that state, while we stepped up our detailed work elsewhere.

My examination of published statements about coal reserves together with our own experience, has clearly demonstrated the fact that a reserve statement is of little value unless a considerable amount of data about how limits of coal are established at depth, minimum thickness of coal considered, weight of coal assumed, and similar factors are provided to show how the estimate was prepared. I am surprised at the number of estimates that do not give this information in full, and at the number of writers and speakers on the subject who attempt to compare estimates made on entirely different bases. Mr. McKay's new estimate of Canadian reserves, which was made most carefully and conservatively, and which includes only the thicker beds currently considered to be minable, cannot be compared with Mr. Campbell's estimate of the coal in the United States, which includes bituminous coal 14 in. or thicker to a depth of 3000 ft. Nor can an engineer's realistic appraisal of coal reserves in beds 4 ft thick or greater within 2 miles of the outcrop, be compared with more generous estimates of total coal in thinner beds and to greater depths. Yet I have seen the published data abused in this manner.

Considerations for Current Survey

In our current work on coal reserves we have had to face several problems peculiar, more or less, to this country. We felt impelled to maintain the 14 in. cutoff established by Mr. Campbell so that new state estimates would be comparable to the older estimates, and so that our tabulation would be complete. It was necessary, therefore, to establish several intermediate cutoff thicknesses, so that data would be available for beds of present economic interest. For the same reason we wished to maintain 3000 ft as the limit of overburden under which mining can be carried out, though obviously, values for intermediate ranges of overburden would have to be calculated. Finally, we had to evolve systematic methods for handling the vast amount of outcrop data available in certain areas where little mining or drilling has been carried on, and where information about coal at depth is not available. With these few technical restrictions we evolved, with help from Mr. Toenges and Mr. Turnbull of the Bureau of Mines, a series of procedures for calculation and tabulation, which we have been following, and which I should like to present at this time for your consideration and comment. Although systematic procedures, and seemingly arbitrary definitions, are necessary, it should be noted that these are guides only, and must be so used. Actually, geologic interpretation and sound judgment enter into the application of these definitions and procedures to a very considerable extent, as is true of all statements of policy and objectives.

Definition of Terms

First, we adapted the standard Geological Survey-Bureau of Mines nomenclature for reporting reserves of mineral commodities to fit the special case of coal. This permitted us to define three classes of coal reserves according to the reliability of data available for making calculations.

MEASURED COAL

"Measured" coal, we say, is coal for which tonnage is computed from dimensions revealed in outcrops, trenches, mine workings, and drill holes. The points of observation and measurement are so closely spaced, and the thickness and extent of the coal is so well defined that the computed tonnage is judged to be accurate within 20 pct or less of the true tonnage. Although the spacing of the points of observation necessary to demonstrate continuity of coal will vary in different regions according to the habit of the coal beds, we are assuming that for "measured" coal the points of observation will be, in general, on the order of ½ mile apart. The outer limit of a block of "measured" coal, therefore, is of the order of ¼ mile from the last point of positive information (that is, roughly one half the distance between points of observation).

Where no data are available other than measurements along the outcrop, but where the continuity of the outcrop is measured in miles, and suggests the presence of coal at great distances in from the outcrop, we feel that a certain amount of coal should be classified as measured. We are proposing, therefore, a smooth line drawn roughly ½ mile in from the outcrop, be used to mark the limit under cover of a block of coal that can be classed as "measured."

INDICATED COAL

"Indicated" coal is coal for which tonnage is computed partly from specific measurements, and partly from projection of visible data for a reasonable distance on geologic evidence. In general, the points of observation are of the order of 1 mile apart, but may be as much as 1½ miles for beds of known geologic continuity. For example, if drilling on ½ mile centers has proved up a block of "measured" coal of fairly uniform thickness and extent, the area of "measured" coal as defined above is, according to the judgment of the estimator, larger than the actual area of drilling by as much as ¼ mile on all sides. If from geologic evidence the bed is believed to have greater continuity, this area of "measured" coal is surrounded by a belt of "indicated" coal, which according to the judgment of the appraiser, may be as much as 1½ miles wide.

Where there are no data available other than measurements along the outcrops, but where the continuity of the outcrop is measured in miles and suggests the presence of coal at great distances in from the outcrop, we propose that two lines drawn roughly parallel to the outcrop, one ½ mile in from the outcrop, and one 2 miles in from the outcrop, be used to define a block of coal that may be classed as "indicated."

INFERRED COAL

"Inferred" coal is coal for which quantitative estimates are based largely on broad knowledge of the geologic character of the bed, or region, and for which there are few, if

any measurements. The estimates are based on an assumed continuity, for which there is geologic evidence. In general, inferred coal lies outside the limits defined for "measured" and "indicated" coal. Where there are only outcrop data on which to base estimates. "inferred" coal is coal lying more than 2 miles in from the outcrop and within limited areas established as I shall describe in a later paragraph.

PROVED, PROBABLE, AND POSSIBLE

It has been suggested that the terms "proved," "probable," and "possible" are in more general use in the coal industry for reporting coal reserves. As I see it, the two sets of terms are loosely comparable, and those who wish to do so might substitute, with perhaps slight revision of the definitions, the terms "proved," "probable," and "possible." The important thing is that such terms be defined.

Methods of Estimating Extent of Coal Beds

We have had to use several ways to limit the assumed extent of each coal bed in preparing our estimates. Where the continuity of the bed is well established by maps of the outcrop, mine workings, and drill holes as in the Pittsburgh bed, the entire area of known occurrence is taken, even though points of observation are widely spaced. Persistent beds that have been traced around a basin or spur are considered to underlie the area enclosed by the outcrop. Otherwise, the length of outcrop within the thickness limits listed above is considered to establish the presence of coal of all classes in a semicircular area having a radius equal to ½ the length of the outcrop. The total area of coal is considered to extend beyond such a semicircle only if mine working or drill holes so indicate; in which case coal is considered to extend only 1 mile beyond the limits of the positive information. An isolated drill hole too far removed to be incorporated in the area thus defined, is considered to determine an area of coal of all classes extending for a radius of ½ mile around the hole.

ESTIMATING WEIGHT OF COAL RESERVES

Many estimates of coal reserves have been based on assumed weights of coal somewhat lighter than available specific gravity determinations indicate. Assumed values as low as 1500 and 1688 tons per acre foot for bituminous coal have been used, for example, in detailed published statements, whereas the average weight of bituminous coal in the ground is nearer 1800 tons per acre foot. We have established, therefore, the arbitrary policy that where other precise data are not available the following values shall be assigned:

	Short Tons per Acre Foot
Anthracite and semianthracite	2,000
Bituminous coal	1,800
Subbituminous coal	1,770
Lignite	1,750

DETERMINATION OF OVERBURDEN

We are working now to provide in our future coal reserves work, very detailed information about the overburden on each category of coal reserves. (By overburden I mean the thickness of rock overlying the coal, regardless of whether it is above or below drainage.) In our work on Montana, however, where topographic information is not always available, we have had to content ourselves with reporting coal between 0 and 2000 ft. As coal is frequently mined under an overburden of more than 1000 ft, and as very little is mined under an overburden of more than 2000 ft, this seemed a reasonable and convenient cutoff, and I wonder if this limit would not be satisfactory in reporting all coal reserves. In most states, however, it will be possible to calculate reserves between 0 and 1000; 1000 to 2000; and 2000 to 3000 ft overburden, and we are prepared to do so.

THICKNESS CATEGORIES

There is some doubt in my mind as to the most desirable thickness categories for reporting coal reserves. At present, in our work on Montana, we are using the following:

For bituminous coal

1. More than 36 in.
2. 24 to 36 in.
3. 14 to 24 in.

For subbituminous coal and lignite

1. More than 10 ft
2. 5 to 10 ft
3. 2½ to 5 ft

It has been suggested, however, that in the future we use the following for bituminous coal:

1. More than 42 in.
2. 28 to 42 in.
3. 14 to 28 in.

I would be very interested to hear expressions of opinion on this most important point.

CALCULATIONS FOR INDIVIDUAL BEDS AND SMALL UNIT AREAS

Finally, we have established to our own satisfaction that it is desirable in calculating reserves to make calculations for individual beds and for small unit areas. In this way the information is of maximum usefulness, and the calculations can be checked and changed as new data become available. We also feel that the legitimate function of the Geological Survey is to compute the total coal in the ground before mining began. This figure is not subject to change because of economic conditions, or changes in mining practice, and it is the base from which coal "mined, and lost in mining" and "recoverable coal" can be obtained by the application of engineering principles.

Conclusion

The coal fields of the United States are large in all dimensions. They cover roughly 350,000 square miles, or approximately one-ninth of the total area of the United States. The coal-bearing rocks commonly are several thousand feet thick, and, as in West Virginia, contain as many as 117 named and correlated coal beds. The thing that impresses me as I watch the progress of our work, is that an estimate of the coal reserves in this great volume of rock is an expensive and time-consuming job. It requires that data be assembled, analyzed, and then presented in numerous categories. Many insidious errors are inherent in the process. People who talk about a quick estimate to "show the facts" are making the error of thinking that because there is an abundance of information, it is easy to make an estimate. Actually, the reverse is true. The more information, the more work required to assimilate it, and the more detail necessary to present it fairly. The "facts" as I see them are that we need in this country a new, detailed estimate of coal reserves, and that to get it will

require both time and money. Any attempt to secure a quick answer will yield a figure that very likely cannot be substantiated, and certainly will not yield information in the detailed form now desired.

It will be about 10 years before a completely new estimate can be prepared for the coal reserves of the United States at our present rate of progress, though of course, much information will be available in the interim. We would like to present such information as we obtain in the form most acceptable to the industry, and to that end we are working closely with coal resource committees of the AIME, the National Bituminous Coal Advisory Council, and the Bureau of Mines.

DISCUSSION

G. H. Cady*—Mr. Averitt's description of the nature of the coal resources investigations of the United States Geological Survey and the progress of this work is very timely in view of the general interest in the subject of coal resources. The Federal Survey is obviously taking the "long view" with respect to the appraisal of the coal resources as part of the preparation of the general geological map of the country and the attendant determination of the quantity of all mineral and fuel resources. It is apparently for regional surveys of the conventional type that the detailed procedures employed by the Federal Survey are applicable. Even if so it is somewhat unexpected to find the same criteria for evaluation will be applied to the country as a whole.

The work on coal resources appraisal has been proceeding at various rates in a number of coal producing states for various periods. Undoubtedly since there has been no common practice in the method of appraisal the states each have worked independently and independently of the Federal Survey, and each has adopted certain practices to meet local conditions. I do not know how much consideration has been given by the U. S. Geological Survey to these practices. It seems probable that such consideration might result in important modifications of the standards which apparently have been set up by the Federal group in line with local requirements.

It is the practice in some state surveys to present the facts in regard to the occurrence and distribution of coal beds and their variations in thickness down to thicknesses of 1 to 1½ ft in relatively narrow stages up to maximum thickness. It is then possible for any one using such figures to compile quantitative estimates of workable coal at various minimum thickness limits. This is a desirable objective where surveys are detailed and information correspondingly good. However, vast quantities of coal lie in beds that are relatively thin about which little information is available and the quantity of such coal might affect the final appraisal to the extent of several thousand million, depending upon the minimum limitation observed. Thus in the case of Illinois a difference of 1 ft in average thickness of coal in the coal field would make a difference of approximately 35 billion tons in the estimated quantity present. One is dealing with a very large area and small differences in thicknesses are very important in the overall picture.

The estimate of 10 years to accomplish the sort of detailed appraisal outlined by Mr. Averitt is viewed somewhat skeptically and certainly would require many more trained geologists than seem to be available. The requisite detailed mapping and study are very time consuming not only because of the geological field work required, which is difficult in certain seasons, but also in order to assemble data from company and personal files, and to study drill records and cores, and to sample and analyze the coal. Unless the staff of geologists could be increased in the order of several hundred per cent the prospects are that the appraisal of the type suggested will require many decades not just one. The permanent value of such appraisal, however, is not questioned and it should be expedited as rapidly as means allow.

Satisfactory appraisal is not easily achieved and will vary from time to time. The main function of a geological survey is to collect and present the facts that will make possible appraisal in terms of existing conditions by individuals, particularly engineers, qualified by training and experience for making such estimates. Thickness and character of the coal bed, nature of the overlying and underlying strata, position, thickness, and character of "partings," relationship to other coal beds, and a variety of other factors must be assembled by the geologist in order to present a complete picture of the geological setting. The economic and engineering factors are in general outside the field of geological experience. The geologist should see to it that the pertinent geological facts are available; much more information than simply thickness and depth of the coal beds is essential to meet all the requirements of appraisal in the future as well as at present.

With respect to the present demand by engineers and coal mine operators and others for a rapid re-appraisal of the available coal resources, in the light of existing practices and those in immediate prospect, the Federal procedure of systematic mapping and appraisal is scarcely in line. It will be necessary for those particularly interested in having such an estimate made to establish standards with respect to the various factors involved so that the scope of the geological work can be definitely restricted for the various coal fields. There is little use in setting up general standards since no such standards will be applicable to all fields. Thus it is folly to establish a standard minimum thickness to be generally applicable which is one-half to one-third the minimum minable thickness in some fields. The greatest difficulty in appraisal of the presently actually workable coal bed lies in the variation in the importance placed on roof, floor, structural conditions, bedded impurities, sulphur content and other factors that affect judgment relative to the value of a coal bed. It is doubtful whether any appraisal will have universal approval in view of the variable factors involved.

For this reason if none other, the facts should be carefully and painstakingly assembled after the manner of a systematic geological survey such as that advocated by Mr. Averitt. Only upon the basis of such information can reliable estimates be made in line with selected conditions.

Paul Averitt (*author's reply*)—In this paper I discussed, but did not define as such, two major phases of our activities on coal, which are closely related, and which normally we in the U. S. Geological Survey see as an integrated whole. It is understandable, therefore, that Dr. Cady should assume that my remarks applied only to the current work on coal reserves with which I am directly concerned. In view of Dr. Cady's discussion, however, I think a brief summary of the two phases of U. S. Geological Survey activities on coal will be appropriate.

The long-term function of the U. S. Geological Survey on coal investigations is to make detailed geologic studies, involving the preparation of detailed geologic maps showing the outcrops and correlations of coal beds, direction and dip of coal-bearing rocks, nature and thickness of overburden, location of faults and fold axes, and similar features. Such studies also yield many measured sections of coal and associated strata, which show variations in the thickness of the beds, and the intervals between coal beds. Taken as a whole, detailed geologic studies provide much useful information for mine operators and engineers. As Dr. Cady points out, this type of geologic work is arduous and time consuming, but it provides the basic data that are necessary both in the development of our coal fields, and also in the preparation of coal-reserve estimates, which are discussed in the next paragraph. The U. S.

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Geological Survey and the state surveys have been doing such detailed work for more than 50 years, yet some areas in the United States have never been mapped, and in many others the work is only of a reconnaissance nature. Dr. Cady is correct in his statements to the effect that the detailed mapping to modern standards of all the coal fields in the United States would require a large staff of geologists for many decades. This, then, may be regarded as the continuous, long-term phase of our activities, which has been in progress for more than 50 years, and may still be incomplete 50 years hence.

From time to time in the course of this long-term program of detailed field surveys in the coal-field areas, we have paused to reappraise the coal reserves of the United States on the basis of the available data. Such appraisals were made in 1913, 1928, 1937, and 1943. The interval between 1928 and the present, however, has witnessed both a major depression and a major war, and during this period we were unable to devote the proper amount of time to the revision of the estimates: the estimates made since 1928, therefore, are dependent in large part on old data. We are now engaged in a new appraisal of coal reserves, based on data currently available, and augmented locally by data from our active program of detailed geologic studies mentioned above. This new estimate, which is intended to be somewhat more detailed and thorough than previous estimates, will nevertheless still be only a provisional estimate, and will include much coal that can be classed only as "inferred" or "possible" because of the paucity of information. Inevitably, such an estimate, like all reserve estimates, will be subject to modification the day after it is prepared. I am sure Dr. Cady will agree that a new estimate of reserves can be prepared on this basis in a modest length of time, depending, of course, on the size of the staff so employed.

Dr. Cady's comment about the possible value to the U. S. Geological Survey of procedures employed by the several state surveys that have estimated their reserves is, of course, most appropriate. I examined with profit the procedures employed by the Pennsylvania, West Virginia, Ohio, Illinois, Kansas, and Canadian surveys, before formulating our own. I also submitted our method of procedure to the Committee on World Coal Resources of the AIME, of which Dr. Cady is a member, and received several very helpful suggestions, which I here take opportunity to acknowledge. Of the procedures employed by other geological agencies, however, only the Kansas and Canadian surveys attempted a breakdown of the data by classes according to the reliability of the information, and none have published a breakdown by range of thickness of beds, though all have provided much data on the range of thickness and average thickness of the individual beds. I believe, therefore, that having profited by our own past experience in estimating reserves, and having borrowed what we could from our friends, we are on the way toward a new reserve figure that will be an improvement on previous estimates, and will present data in the detail required for modern needs. It is our hope, of course, that all state surveys will collaborate on coal-reserve estimates to some standard specifications. Some will doubtless be able to contribute more in data and evaluation than others, but all should participate. Naturally we expect to arrive at mutual understandings with the state surveys regarding the scope of responsibilities to be assumed.

I agree with Dr. Cady that the main function of a geological survey is to collect and present the facts, because the fundamental basis for all coal-reserve estimates is furnished by geologic surveying of coal outcrops, the establishment of the correlation of the coal beds between points of observation, and the measurement of bed thicknesses. It is likewise a function of the geologist to estimate the reserves of coal in the ground, which calculations can be made directly from the geologic data, and other data obtained during the course of field surveys. Naturally, studies of recoverability and minability are the functions of mining engineers. It is, however, the duty of geologists to present their data on reserves in the form that will be most useful to aid in studies of minability, and to aid those who would mine the coal, or make plans based upon its minability. It is for this reason that we are now making reserve estimates in several classes, both according to the reliability of the data on which the estimate is based, and according to the range of thickness of the beds.

The need for a breakdown of coal-reserve data into classes will materially increase the time required to do the job, but this is an inevitable consequence of the demand for more detailed information. A better job always takes longer, or takes more men. The U. S. Geological Survey can complete the coal-reserve appraisal in less time if larger funds are made available.

The establishment of standards and procedures for carrying on coal reserves work is one of the most difficult problems to resolve. As Dr. Cady points out, "In the case of Illinois, a difference of 1 ft in the average thickness of coal in the coal field would make a difference of approximately 35 billion tons in the estimated quantity present." It is obvious that coal reserves no longer can be estimated on so gross a scale. It is obvious also, that some procedures must be established for carrying on the work. Some simple, reasonable procedure is required, for example, in weighting the average thickness of coal used in making the estimate for each individual bed. Likewise, there must be some simple, reasonable procedure for defining the area of coal to which a given thickness is applicable. Otherwise, large gross errors of overestimation will be perpetrated.

The principles we have established have evolved slowly as new problems have arisen and new suggestions have been received. We have, for example, accepted the recommendation of the National Bituminous Coal Advisory Council that reserves be calculated for bituminous coal in the thickness range between 28 and 42 in., and if possible, at 1 ft intervals above 42 in. We have also accepted the recommendation that the reserves be calculated for the several subdivisions of the bituminous rank, and, wherever possible, for sulphur content in certain ranges. Dr. Cady's statement that "There is little use in setting up general standards . . ." reflects, I think, his long experience in coal reserves work, and his awareness of the magnitude of the problem of calculating reserves to fit every specification and every set of mining conditions. We, too, are aware of this difficulty. The problem should be faced, however, not by abandoning standards and procedures, but by establishing a very few simple standards and procedures that seem to fit most needs, and apply to most regions. This we are trying to do. In practice, the standards and procedures that we have evolved, and that are still changing, are used merely as a general guide to establish a reasonable uniformity of thinking on the part of the geologists engaged in estimating reserves. It is clearly necessary for each geologist to rely heavily on his own judgment, based on his knowledge of the habit of the coal and associated rocks in the area being appraised. As I have stated, "Although systematic procedures and seemingly arbitrary definitions are necessary, it should be noted that these are guides only, and must be so used. Actually geologic interpretation and sound judgment enter into the application of these definitions and procedures to a very considerable extent, as is true of all statements of policy and objectives."

The current interest in coal as a potential source of liquid fuels, and the insistent demand for precise quantitative information about our coal reserves should be very gratifying to many members of the coal industry who have pointed out the need for this very thing for so many years. I think now that with the support of industry and with the application of considerable labor the U. S. Geological Survey and the state surveys acting together can produce during the next few years a new estimate of coal reserves that will serve current needs.

History of Pumping at the Chief Consolidated Mine, Eureka, Juab County, Utah

By JOHN G. HALL,* Member AIME

The pumping operations at the Chief mine have been unique in the respect that for many years the entire flow of water into the mine has been disposed of by pumping into natural underground "caverns" or "sinks." Volumes up to 4000 gpm have been pumped into these caverns with no apparent tendency for the caverns to fill with water, and it has never been proven that any of the water returns to the mine workings.

Major pumping operations have been carried on during two periods, the first from 1918 to 1927, and the second from 1942 to the present time.

Early Pumping Operations

CHIEF MINE

The Chief Consolidated Mining Co. was organized in 1909, and for the following nine years ore was developed and mined from the 600 level down to the 1800 level, all above the permanent water table, the 1820 level. The ore occurs as extremely-irregular pipes of replacement ore in limestone, and to date a total of approximately 2½ million tons of ore has been produced. The ore was highly oxidized down to the permanent water level, and some highly-oxidized ore runs occur beneath the permanent water table.

In 1916, with extensive mining being done on the 1800 level, 20 ft above the water table, there were indications that a zone of secondary enrichment might exist below the water, and in order to follow the ore runs to greater depth, it was realized that arrangements would have to be made to pump water to the surface.

The original pumping installation was designed to pump 350 gpm in five stages against a 1900-ft head to the surface. By January 1919, the water had been lowered to the 1900 level and an excellent grade of silver-lead ore was being produced from the zone of secondary enrichment. At this point, the flow of water equalled the capacity of the pumps, and pumps of an additional 1000 gpm were installed to supplement the original system. By January 1920, the water had been lowered approximately 100 ft, and the total flow of water was 1000 gpm. The water was being pumped from many different places in the mine where ore was being followed downward, with 27 different pumps operating below the 1800 level, mainly Cameron air-driven sinker pumps. These first sinking pumps were cumbersome and inefficient, and each of the larger pumps weighed approximately three tons and required tremendous volumes of compressed air.

During this period, an exploration drift being extended north and west from the Chief shaft on the 1600 level encountered a large natural cavern some 2000 ft from the shaft. This cavern in places measured 60 ft in diameter. Sometime later the disposing of water into this cavern was suggested, and as an experiment to determine the results of such an operation, an attempt was made to fill the cavern by pumping in a small volume of water, but the water disappeared as fast as it was discharged into the cavern. It was decided then to try the cavern as a drainage outlet for the entire volume of water being pumped to the surface, and the water apparently did not return to the mine workings. Subsequently water up to 3000 gpm was pumped into this cavern. An attempt was made to determine whether any water was being returned from the cavern to the lower workings, and 100 lb of sodium fluorescein salt was added to the water discharged into the cavern, but no trace of it was found in the drainage water in the mine. Results of another such test will be described later.

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TP 2613 A. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Aug. 30, 1949. Manuscript received Jan. 10, 1949.

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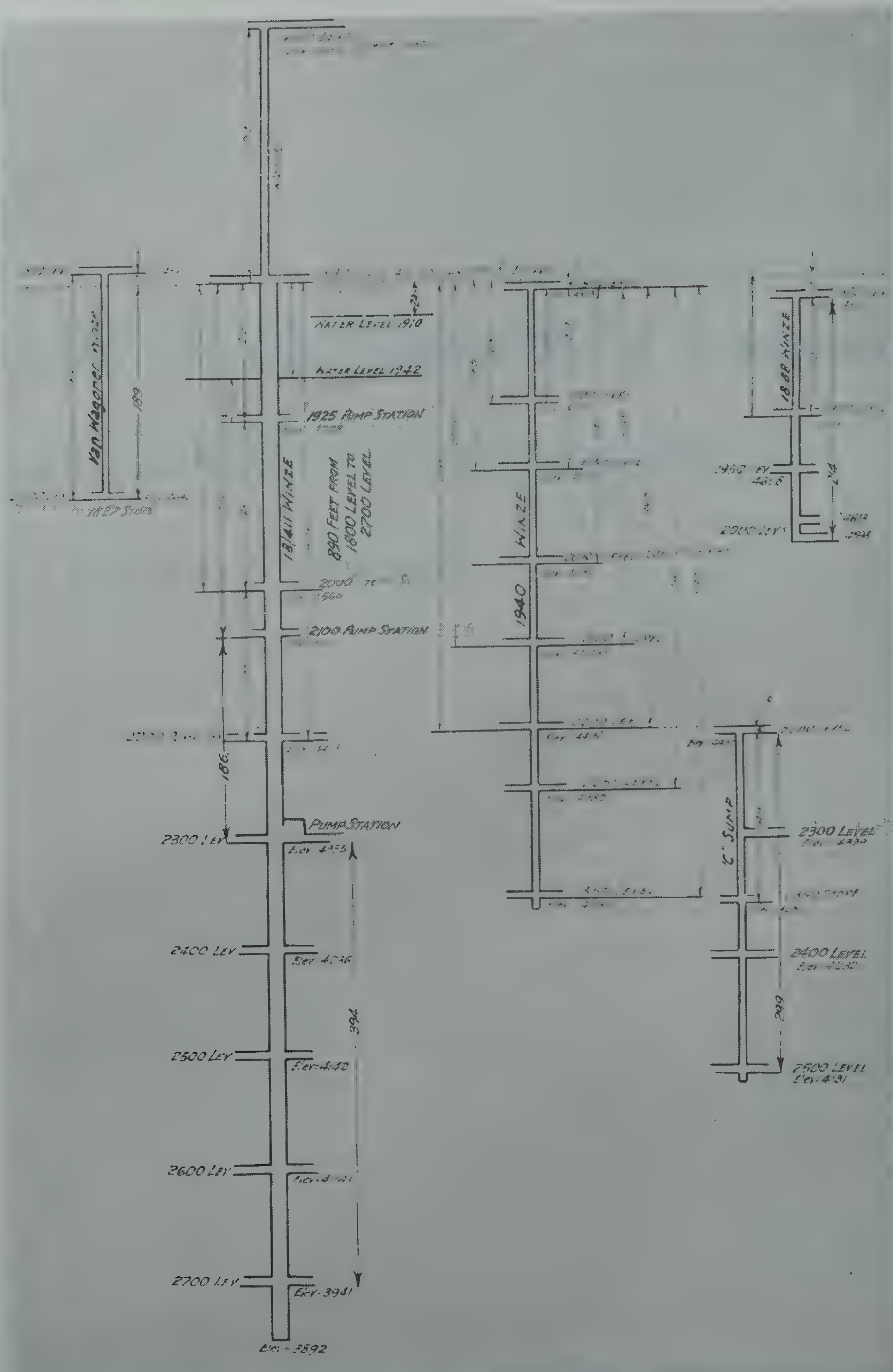

FIG 1—Generalized section of main winzes operating below the water level at the Chief Consolidated No. 1 mine.

By 1921, ore had been followed down to the 2000 level and two single-compartment vertical winzes had been sunk to the 1900 level (18-88 and 1800 winzes), and about this time another single-compartment shaft was raised from the 1900 level to the 1800 level. This is now known as 1940 winze. Fig 1 shows a generalized section of the main winzes.

With the success thus far encountered by discharging the water into the 1600 cavern, it was decided to attempt to dispose of the water pumped from 1940 winze somewhere on the 1800 level to eliminate the 200-ft lift to the 1600 cavern. With this in mind, a pipeline was run some 2000 ft to the bottom of the Chief No. 2 shaft on the 1800 level. It was hoped that this area would contain a crack or opening that would absorb the water pumped into it. However, the ground in this area proved to be very impervious, and the experiment met with no success. Later, a natural cave was encountered 1000 ft north of 1940 winze and an attempt was made to pump into this. A cement bulkhead was constructed across the drift several hundred feet from the cave and water was pumped into it. A pressure gauge was installed in the bulkhead and within 12 hr the pressure behind the bulkhead had built up to 175 lb. This attempt to dispose of the water was abandoned.

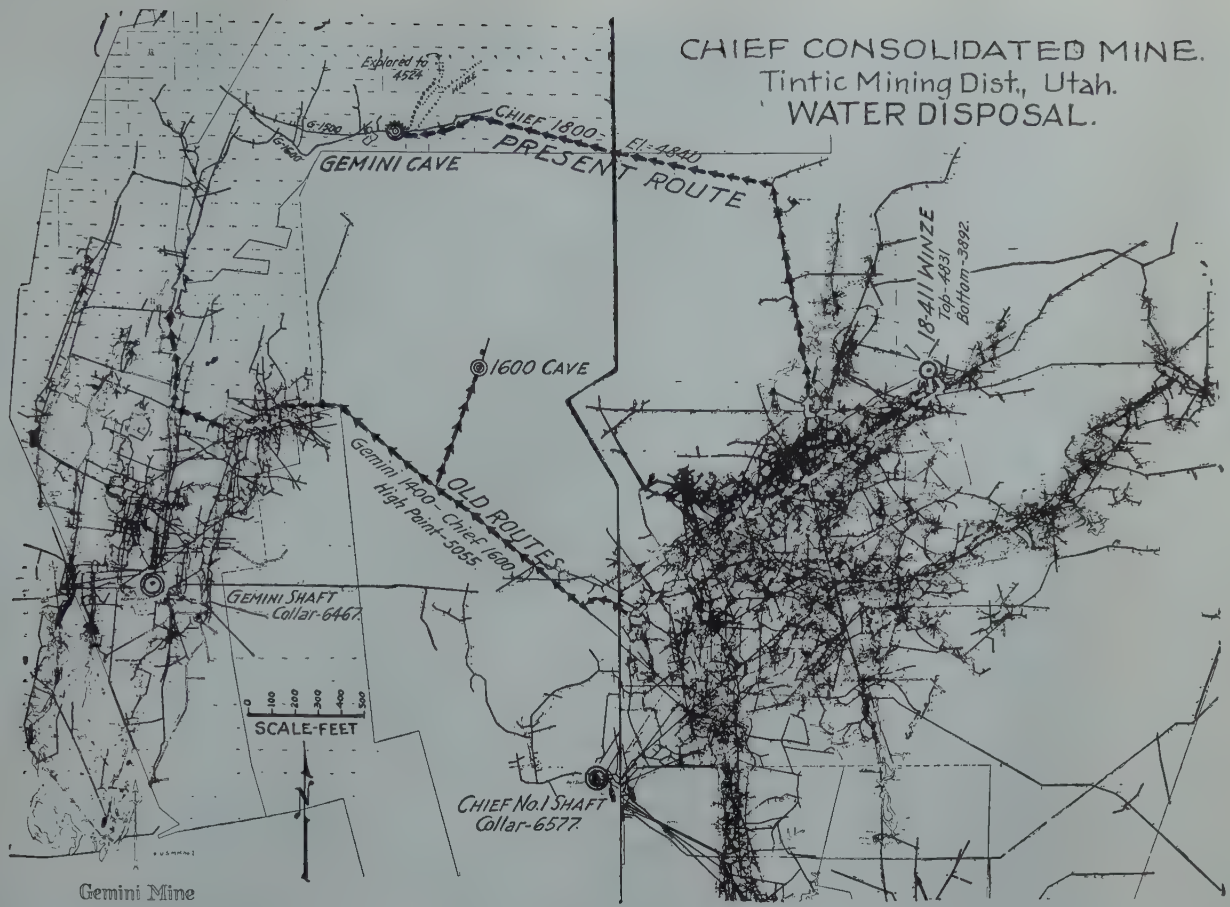

FIG 2—Composite plan map of Chief No. 1 mine and Gemini mine showing the locations of 18–411 winze, the 1600 cavern and the Gemini cavern. Also shown are the old and present routes to the caverns.

By 1924, it was felt that the underwater ore bodies were large and continuous enough to warrant sinking a three-compartment vertical shaft in limestone from the 1800 level to the 2200 level to facilitate mining and pumping operations. This shaft is now known as "18-411 winze." When this winze was completed to the 2200 level, a new and much more efficient pumping system was installed to replace the old system that had been built up gradually over a 6 year period to cope with changing conditions as workings were extended ever deeper and the volume of water increased. A raise was driven from the 1800 level, at 18-411 winze, to connect to the 1600 level to accommodate the pump lines. The water was then piped from this point directly to the 1600 cavern some 1900 ft westerly of 18-411. The newly-installed pumping system was so designed that all the water from the mine could be collected in a sump on the 2200 level, and from this point was pumped in three lifts by series-connected centrifugal pumps directly to the 1600 cavern. This system had a capacity of 1700 gpm, with a standby system of the same capacity. Fig 2 is a plan of the water disposal system.

From 1924 to 1927, workings in the 1940 area were extended to the 2350 level and another subshaft known as "C" sump was sunk on or near ore from the 2200 level to the 2500 level which was the deepest level reached.

In March of 1927, a combination of unexpectedly adverse conditions occurred which affected the profitable operation of the Chief mine. Foremost of these was the drop in metal prices. To meet these adverse conditions, it was deemed expedient to give up the lower workings, and the pumping expense that was involved in their operation. It was planned to attempt to follow additional ore runs down to the water level to help pay the expense of any future pumping operations.

When pump operations were stopped, the pumps, all electrical equipment, and mining tools were removed from the lower levels, but all pipe and track was left intact. The lowest levels flooded rapidly, and the water rose from the 2500 level to the 2200 level in about 24 hr, from the 2200 to the 2100 in two days, and from the 2100 to the 2000 level in about three weeks. From this point the water rose very slowly and 24 months later the water was at the 1950 level. Twelve years later, in 1942, the water was still only up to the 1860 level or 40 ft below the original level, although it was rising gradually still.

Developments at the Chief mine from 1918 to 1927 had carried operations 700 ft below the natural water table. Beginning with a few gallons, requirements increased to a maximum of 2000 gpm when operations ceased.

An interesting sidelight concerning the 1600 water cavern is the fact that when pumping was halted in 1927 it was found that a considerable accumulation of slime material had settled out of the pump water in the drift approaching the 1600 cavern. Assays were made upon this material and as a result some 480 tons of ore averaging $7.65, at 1928 prices, was retrieved. This material had been carried in the drainage water during the period of pumping.

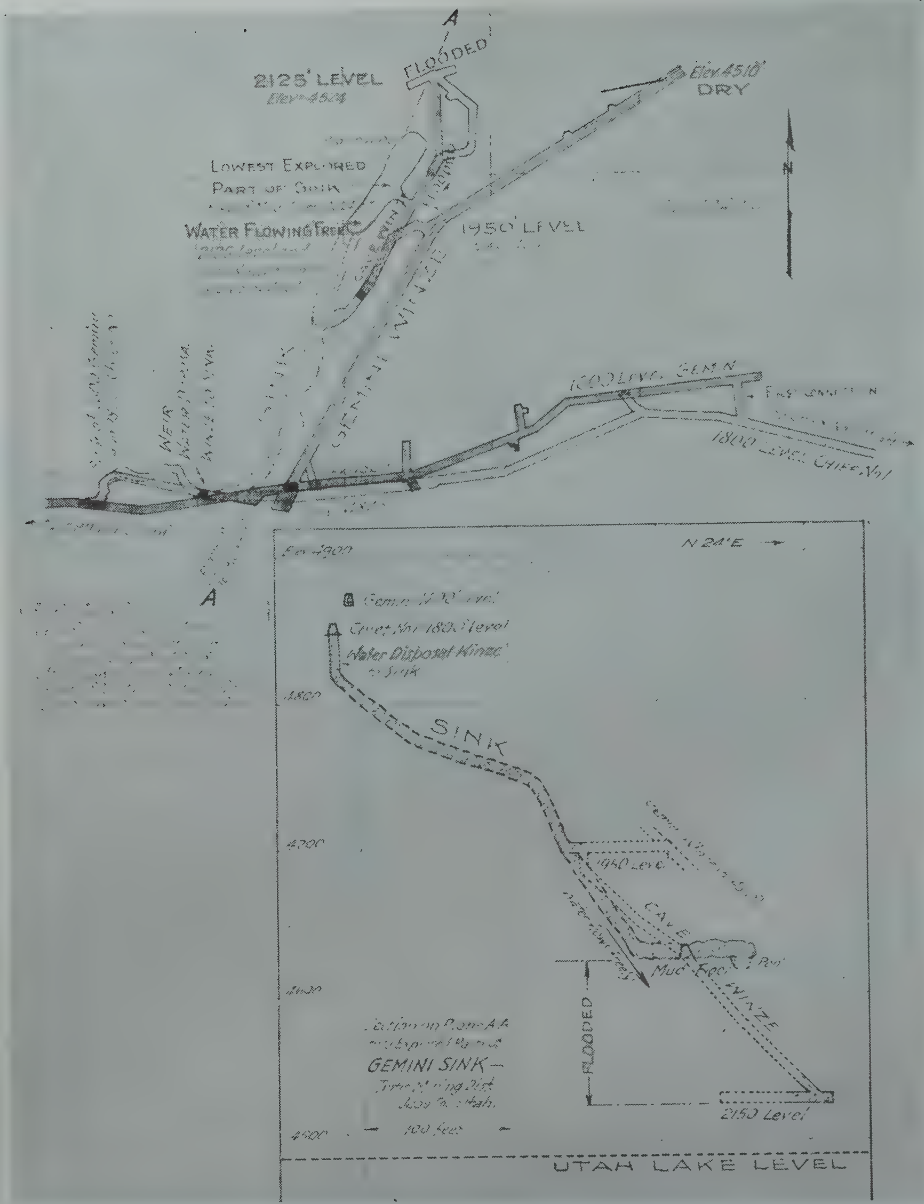

FIG 3—Plan map and section showing details of workings in and near the Gemini cavern, Chief Consolidated mine.

GEMINI MINE

During the period 1918 to 1925, pumping operations were also carried on at the Chief Consolidated Mining Company's Gemini mine. An exploration drift on the Gemini 1600 level, which approximately equals the Chief 1800 level, also encountered a natural cavern, or sink, some 1100 ft northwesterly from the Chief 1600 cavern. With the success of the Chief 1600 cavern it was decided to pump the Gemini water into the Gemini cavern rather than to the surface, and this operation also met with success. The total flow pumped into this cavern at that time was only 300 to 400 gpm, although this cavern now assumes an important role in Chief pumping.

The Gemini cavern is some 1600 ft northwest of the 18-411 winze in the Chief mine, and when pumping was halted at the Gemini in 1925, it was decided to attempt to explore and survey its downward extensions. It was hoped that it might be possible to follow the cavern downward to an elevation lower than the water level in the Chief, and then to drive a drift from the Chief to connect with the cavern, and thereby greatly reduce the head requirements of the pumping system. Two men first descended into the cavern on ropes, and were successful in exploring the flat-lying cavern some 200 ft of vertical elevation below the Gemini 1600 level or approximately 160 ft below the natural water table. At this point a large room with several feet of mud on its floor was encountered. No opening large enough to allow the men to descend farther could be found leading out of this room. It was then decided that in order to further explore the cavern it would be necessary to sink a small incline winze near the cavern, and to keep contact with the cavern by drifting to it at various elevations. The winze was extended to the elevation of the mud floor and some 100 ft below, but no large openings were found below this point, although the limestone was extremely vuggy and porous. Explora-

tion of the cavern was abandoned shortly before the pumping was halted in the Chief mine in 1927. Fig 3 shows plan of workings at Gemini cavern.

From 1928 until 1942 mining operations at the Chief mines were confined to the areas above the water level except for one brief attempt to follow ore below the water. This was in 1941, when a group of block leasers followed the extension of an ore run known as "Millionaires' Row" down to the water level, which was still below the 1900 level (1950 level at 18-27 winze). An attempt was made to follow this ore run below the water by pumping the water from this area to the 1800 level, and thence into 18-411 winze some 900 ft to the west. The water in this area was lowered 50 ft by the lessees; however, with obvious recirculation occurring, and with very limited pumping capacity, it became necessary for them to abandon this attempt.

Pumping Operations 1942 to 1946

Early in 1942 it was announced that because of the critical shortage of lead and zinc, the government would allot to mining companies an increased price for lead and zinc based on a tonnage quota to be determined from 1941 production. Any metals produced over and above this quota would be paid for at the advanced price. It was known that the underwater areas of the Chief mine contained large unmined tonnages of these products developed during the period 1918 to 1927. During this early period of mining, the zinc contained in the ores brought penalties from the ore buyers, and therefore mining was done as selectively as possible, and ore with high zinc content was left unmined. Also, metal prices were low and mechanical mining equipment was practically lacking in 1927 when pumping had been stopped.

It would now be possible, by shipping these lead-zinc ores to custom flotation mills, to mine the ore runs on a large tonnage basis by replacing the expensive square-setting methods by cheaper open-stoping and shrinkage methods. Also the use of modern mechanical mining equipment would further decrease the mining costs from the 1918 to 1927 level. It was therefore decided to unwater the lower levels of the Chief mine.

The original scheme was to unwater through 18-411 winze to the 2200 level, the bottom level of the shaft at this time, and then further to unwater through "C" sump subshaft from the 2200 level to the 2500 level, the lowest level of the mine. The difficulty of obtaining the proper pumping equipment, because of wartime scarcities, necessitated using equipment immediately available to best serve the purpose. The close cooperation of the Park-Utah Mining Co. and the diligent work of the late Leonard Wilson, Consulting Engineer for the Chief, greatly speeded the unwatering operation.

It was decided to discharge the water into the Gemini cavern, rather than into the 1600 Chief cavern, by pumping the water to the 1600 Chief level, as before. This time the water was allowed to flow by gravity through a drift connecting with the Gemini mine, and thence with the Gemini cavern. This cavern was used rather than the old 1600 Chief cavern, for it was felt that if there were any possibility of some portion of the water returning to the mine, that a cavern 1100 ft farther from the mine workings would return less water.

Unwatering operations were started in August 1942. Deep-well and horizontal centrifugal pumps with a capacity for discharging 2400 gpm to the Gemini cavern were installed at the 1800 level collar of 18-411 winze. The water was lowered to the old 1925 pump station level with these pumps. The next step was to lower a 350 hp, 3000 gpm Byron-Jackson submersible-type pump down 18-411 winze from the 1925 level to the 2150 level, and to install a booster pump on the 1925 level. With this system the water was lowered to the 2150 level. In the installation of this pump, which was lowered on its column beneath the water in 18-411 winze, it was found that the shaft was completely blocked at about the 2150 level. Therefore, it was necessary to further unwater through 1940 shaft.

In May 1944, with the water standing at the 2150 level, a project was completed that would simplify, expedite, and lower the cost of unwatering the lower levels of the Chief, and for all time reduce pumping costs at the Chief. A drift was connected on the Chief 1800 level directly with the Gemini cavern so that the pumped water could be discharged into the cavern without the necessity of lifting it 215 ft higher to the 1600 level. With the pumps operating at this reduced head, the volume pumped was increased to approximately 4000 gdm, and the water level was lowered rapiply after this date.

At this time it was decided to explore again the Gemini cavern with the thought in mind of connecting with the cavern on a level lower than the 1800 level. The intention was to diamond drill from the old winze to explore for the cavern below the mud floor. When the diamond drill hole had been extended a short distance, water was encountered and the drilling halted. The mud floor was explored again and it was found to be flooded with water. As an experiment, the pumps were shut off for a short period of time, and the water then receded from this mud floor. When the pumps were started again, the water slowly rose to its former level. The level of the water flooding the floor seemed to have no relation to the amount of water, up to 4000 gpm, discharged into the cavern. It was felt that further exploration of the cavern might be unwise, for it would require blasting into the mud floor, and there was a possibility of damming the cavern, and making further water disposal into it impossible. Exploration of the cavern was abandoned again.

During this time, by the use of two 350 hp submersible pumps which were lowered from the 2100 level of 1940 winze to the 2300 level, the mine was unwatered to the 2300 level. The water was piped from 1940 winze along the 2100 level to 18-411 winze where booster pumps lifted the water directly to the cavern. Upon reaching the 2200 level, of 18-411, through 1940 winze, it was found that broken rock had accumulated on the 18-411 winze 2200 station and that this had been the obstruction encountered in lowering the original submersible pump in 18-411 winze. It was also found that the collar of "C" sump subshaft, and the stope above had caved. It was therefore decided to extend 18-411 winze to the 2300 level and to connect to "C" sump on this level and to unwater to the 2500 level from this point. Sinking was completed late in 1945, and the two 350 hp submersible pumps were removed from 1940 shaft and installed in 18-411 winze with the pump intake below the 2300 level in the shaft sump. These pumps were series-connected to booster pumps on the 2100 level. This again established 18-411 shaft as the main pumping area. Early in 1946 the 2300 level was connected to "C" sump, and a 125 hp

submersible pump installed there to unwater to the 2500 level. This completed the unwatering to the lowest level of the Chief mine.

During 1947, 18-411 winze was extended to the 2735 level and stations cut on the 2400, 2500, 2600, and 2700 levels, the present bottom of the mine.

Present Pumping System

The present pumping system consists of two separate sets of pumps each with a capacity of 3000 gpm at 950 ft head. These discharge to the Gemini cavern on the 1800 level. Both sets of pumps are automatically controlled from a central control panel on the 2300 level. The automatic features control the water level in the sumps, start pumps after power failures, and change pump systems in the event of failure of the operating system.

The No. 1 System consists of a 350 hp submersible unit in the shaft sump on the 2700 level which pumps to a sump on the 2300 level. A 500 hp centrifugal station pump, primed by a 40 hp deepwell pump, lifts the water from this point directly to the Gemini cavern on the 1800 level. This system has a capacity of 3000 gpm. The 500 hp pump is operated by a variable-speed motor, the speed of which is automatically controlled by a liquid rheostat to maintain a constant water level in the 2300 sump.

The standby system consists of a 350 hp submersible unit on the 2700 level, a 350 hp submersible unit in the 2300 sump, series-connected to a variable speed 300 hp booster pump on the 2100 level, which pump directly to the Gemini cavern. The speed of the 2100-level booster pump is controlled by a float switch in the 2300 sump which in turn controls a system of resistors on the 2100 level. This system also has a capacity of 3000 gpm.

During the period from 1942 until 1948 the Chief mine was unwatered from the 1860 level to the 2500 level, and new developments carried the workings an additional 235 ft downward to the 2735 ft level, 915 ft below the original water table. During this period, approximately 500,000 tons of lead-zinc ores were produced. Upon unwatering, the mine was found to be in excellent condition; timbers, pipelines and track were well preserved and in usable condition. In many areas, an accumulation of slimes or mud up to 3 ft in depth had deposited in the drifts. These slimes were found to contain enough lead and zinc to warrant shipping them as ore.

The total flow of water into the mine is now 2600 gpm, which amounts to about 50 tons of water pumped to each ton of ore extracted. The seasons have no apparent effect on the quantity of water entering the mine. The amount of water developed is not proportional to the depth sunk, but is more nearly proportional to the distance opened laterally on the lower levels. Local structures affect both the level and the quantity of water encountered, but the general effect of the pumping is to completely unwater a very flat (4° from horizontal) funnel-shaped area radiating from the pumping point. When the water was lowered 700 ft in the Chief mine, records show that the water was lowered 200 ft in the Grand Central mine, approximately one mile to the south. Little more is known about the effects of the pumping on other nearby areas, as surrounding mines are not developed below the original water table.

The use of submersible-type pumps has greatly reduced the hazards of flooding due to sudden inrushes of water, to prolonged power interruptions, or to other unforseen conditions which might cause pump operations to cease for short periods of time. At the present time, the mine could flood from the 2700 level to the 2300 level without damage to the pumping system. This gives a leeway of several days in which the pumps could cease to operate before flooding, an interesting comparison to the system used in 1927, when the lowest pumps flooded a very few minutes after stopping.

Water Recirculation Test

In December 1948, a test for recirculation of water from the Gemini cavern to the Chief mine was made by the use of sodium fluorescein salt which stands at the top of the list of chemicals and dyes used for tracing underground waters. Fluorescein has a reddish-orange color, but when dissolved in water and diluted, appears a brilliant green. Under ultraviolet light, the water solution fluoresces a brilliant green. One part of fluorescein in 40 million parts of water is visible to the naked eye, one part in 400 million parts of water is easily detected under the ultraviolet light, and with special care, one part in 10 billion parts of water can be detected by the aid of a long glass tube. Eight kilograms of fluorescein salt were added to the water discharged into the cavern, and water samples taken at regular intervals at various points in the mine over a period of several months. No trace of the fluorescein was found in the mine water. The 8 kg of fluorescein added to the discharge water at the cavern is enough to pollute the total amount of water pumped in approximately nine months. Numerous tests were made to determine if contact with any known minerals or rocks in the Tintic District would weaken or destroy the fluorescein color, and nothing was found that would affect the color determinations. It is felt that this test substantiates the belief that water discharged into the Gemini cavern on the Chief 1800 level does not return to the mine.

Theory of the Caverns

There have been many theories offered to explain the disappearance of the water discharged into the Gemini cavern. One theory is that the area being unwatered is a perched water table, and that the cavern is a sealed pipe that extends below this perched water table. This is comparable to pumping from one tank to another some distance below. The water cannot return to the upper tank. The fact remains however, that the mystery of the cavern is yet to be solved.

The use of this cavern 1800 feet below the surface for the disposal of the entire flow of water into the workings of the Chief mine has effected tremendous savings in investments in pumping equipment and costs by reducing the static head requirements of the pumping system from 2735 ft, required to pump to the surface, to 935 ft, to discharge to the cavern. Without this cavern the economical operation of the underwater areas of the Chief would be a question.

Acknowledgment

Appreciation is expressed for helpful information received from previous reports written by the late Walter Fitch, Sr., Cecil Fitch, Sr., H. J. Pitts, and J. Fred Johnson, past and present officers of the company, also to information given by Sam Colovich who worked on the original 350 gpm system installed in 1918, and who is still in charge of the pumps at the mine.

Sublevel Stoping in Small Mines

By J. J. LILLIE,* Member AIME

Sublevel stoping was first developed in the Michigan iron mines many years ago. Since that time this method, and modifications with long hole drilling, have been used in a number of nonferrous mines and have been described in various papers and articles. With a few exceptions, the operations where sublevel methods have been applied are producers of large daily tonnages from ore bodies of considerable size. In selecting mining methods for small properties or relatively small detached ore bodies, sublevel methods often have been overlooked.

The method has been used successfully in relatively small operations at Copper Canyon, Nevada, and, more recently, in mining small ore bodies at Darwin, Calif. As a result of this experience, we believe that a review of the general procedure, the advantages noted, and the results obtained will be of interest to many operators of relatively small mines.

General Description

Briefly, sublevel stoping consists of, first, driving sublevel drifts at predetermined vertical intervals through the length of the stope block; second, breaking an initial mining slot across the stope; third, driving crosscut benches from each sublevel by slabbing into the slot; and, fourth, breaking the ground between benches.

Applicability

Since this is an open stope method, it requires wall rock that will stand over reasonable stope lengths without caving or excessive slabbing. Ore must be moderately firm but need not be hard.

The method is best adapted to steep dipping, tabular or lenticular ore bodies but may be applied to ore bodies of almost any shape if their vertical extent is appreciable. It can be used with dips that are too flat for successful shrinkage stopes and can be modified for use with slushers on dips below about 40°.

Initial mining at the Copper Canyon operation consisted of sublevel stopes for the wider ore sections and shrinkage stopes for the narrow sections where ore widths averaged about 15 ft. After some experience and a study of comparative results on completed stope blocks, sublevels were substituted for shrinkage stopes even in the narrow ore widths.

Practical stope lengths depend upon the character of the walls and the continuity of values within the ore body. Where blocks of low grade or waste are included in the stoping limits, these areas can readily be left as pillars for intermediate wall support and longer stope lengths may be planned. Raises can be spotted in narrow or low grade areas and stope lengths adjusted accordingly.

Stoping Details

To illustrate stoping details, assume that we have an ore shoot 15 to 30 ft wide and about 200 ft long, with walls that would stand over a stope length of 80 to 90 ft (Fig 1).

PRELIMINARY DEVELOPMENT

Preliminary development for sublevel stoping would be simple. A service raise is driven at about the center of the block and stope limits planned to leave a pillar around this raise. Sublevel drifts driven each way from this central raise will outline the

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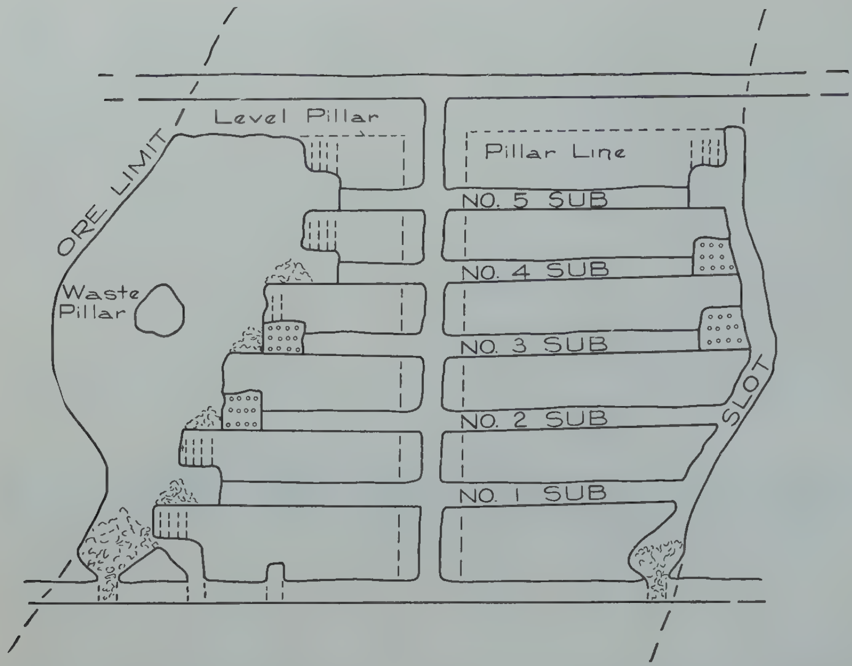

FIG 1—Sublevel stopes: longitudinal section showing general layout and stoping sequence.

end limits of the ore shoot at various elevations. Stoping will start from the ends of the sublevels and retreat toward the central pillar.

HAULAGE LEVEL DEVELOPMENT

On the haulage level, draw points are laid out so that all of the ore can be pulled when broken. The type of chute or draw point selected determines the elevation of the first or lowest sublevel.

At Copper Canyon our first stope layouts used grizzlies over timbered chutes with one or two finger raises feeding each grizzly. After some experimenting we found that the least expensive and generally most satisfactory draw point consisted of a crosscut driven at about a right angle from the haulage drift with a finger raise from the crosscut face.

Ore from the stope piles up on the floor of the crosscut and is picked up with a mucking machine. The length of the crosscut should be planned to allow the mucking machine at least 5 ft of travel between the main line track and the toe of the muck pile. The crosscut track is independent of the main line and the mucker dumps into the side of the cars as the train is pulled past the crosscut. The mucking machine is moved to other points by using a turnsheet on the main line tracks. This side loading eliminates switching and uncoupling of cars and proved faster and more satisfactory than a similar layout with a switch into the crosscut. Oversize boulders are rolled to one side and broken by the mucking machine operator between trips by the train (Fig 2).

Installation, operating and maintenance costs are low with this type of draw point. It has the further advantage, of particular importance in small ore bodies, of reducing to a minimum the amount of ore tied up in a pillar between the haulage level and the bottom of the stope and makes subsequent recovery of this pillar simple.

With any type of draw point, only those at the end of the block where stoping starts need be completed before full production is possible. The rest of the draw points may be prepared as stoping progresses.

FIG 2—Cross section of mucking machine draw point.

Above the haulage level, the finger raises are belled out until connected to form an undercut, which is mined to the first sublevel. The elevation of this lowest sublevel should be planned so that it can be reached with one or two stope rounds drilled from the muck pile over the fingers. The undercut stope need be taken out only fast enough to provide easy pulling of broken ore from above.

SUBLEVELS

Sublevels above the first one are driven from the raise at vertical intervals of about 23 ft. Our first sublevels at Copper Canyon were driven with a large cross section to help produce initial mill ore tonnage. Best overall results were obtained later with small drifts driven on a 5 to 8 pct grade. Drift rounds were drilled with stopers, thus reducing the number of types of machines and steel in service. Mucking was done with slushers, scraping to the service raise chute.

STOPING

Stoping is started at the end of the block opposite the service raise by excavating a slot across the stope over the first finger raises and extending upward to the top sublevel. At first our mining slots were narrow, transverse shrinkage stopes with access from the sublevels. As we gained experience, the slots were started by driving a series of short raises to connect the ends of the sublevels. Since each lift takes only two or three rounds, driving is fast and inexpensive. This raise is then enlarged by benching from the sublevels until a slot from wall to wall is excavated. The enlarged tops of the first fingers form the bottom of this slot.

At each sublevel, benches, or three-sided crosscuts, are driven to both walls by slabbing into the open slot. Benches are usually driven 10 to 12 ft wide and 9 or 10 ft high. The overhang between the top of one bench and the bottom of the bench above is broken by drilling 9 ft holes from below. Vertical spacing of sublevels to permit this procedure eliminates mucking off benches for down holes and allows all stope drilling to be done with one type of drill.

As the benches retreat from the slot we found some advantages in keeping a curved bench rather than a straight one, with the greatest retreat near the center of the stope. These curved benches are more stable and wall support near the active mining areas is better. More confidence in safety is also created since the miner can readily see the condition of the face below his bench (Fig 3).

Blasting from the top sublevel bench was completed first and thereafter each sublevel bench was kept at least one step ahead of the bench below. This sequence is the reverse of more common sublevel practice but sacrifices none of the advantages of the method and adds considerably to safety. Working benches have better structural support and men do not work on overhanging ground. As viewed from the higher benches, the appearance of the stope, with its stepped-back benches, gives the miner confidence in the ground on which he works; a definite advantage with inexperienced underground crews. Some of the broken ore collects on benches but need not be shoveled as this muck pile is dropped into the open stope with the next blast from below.

Fragmentation in breaking the bench and overhang can be closely controlled by hole spacing and blasting practice. Where limitations imposed by haulage, hoisting, or crushing equipment demand close grizzly spacing, this factor is important. In such situations the advantages of low cost primary breakage that produces a high percentage of oversize may be completely offset by excessive secondary breaking costs.

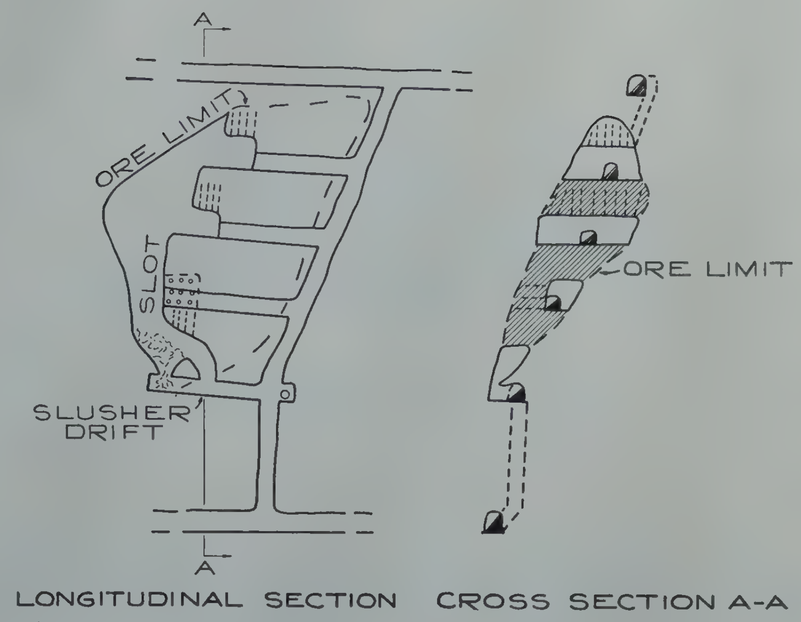

FIG 4—Layout for isolated lens.

Modifications

It is evident from our experience that numerous modifications in general layouts and operating detail will suggest themselves when local conditions are studied in connection with the possibility of sublevel stoping.

Smaller ore bodies may be mined with a service raise at one end, either in ore or just outside the ore limits (Fig 4).

If the ore body bottoms above the haulage level, a slusher drift can be driven just below the ore with finger raises as required. Ore is scraped to a chute at the service raise. This same general layout can be used whenever it is desirable to concentrate loading at one point on the haulage level.

When higher production from an area is desirable, the mining slot can be located in the middle of a block and stopes retreat both ways from this common slot.

In some cases, especially in smaller ore bodies, finger raises may be eliminated and the stope opened directly from the level. Broken ore is slushed to chutes or loading ramps located outside of the stope limits (Fig 5).

The sublevel interval may be varied according to local conditions. In one instance at Copper Canyon, while mining a level pillar along with the stope from below, we broke a 30 ft interval by first taking a normal bench on the top sublevel, then blasting a 5 or 6 ft round out of the back. The remaining ground to the level above was then broken with holes drilled from the muck pile. A larger interval between sublevels with such two stage blasting of the overhang may be advantageous in some cases.

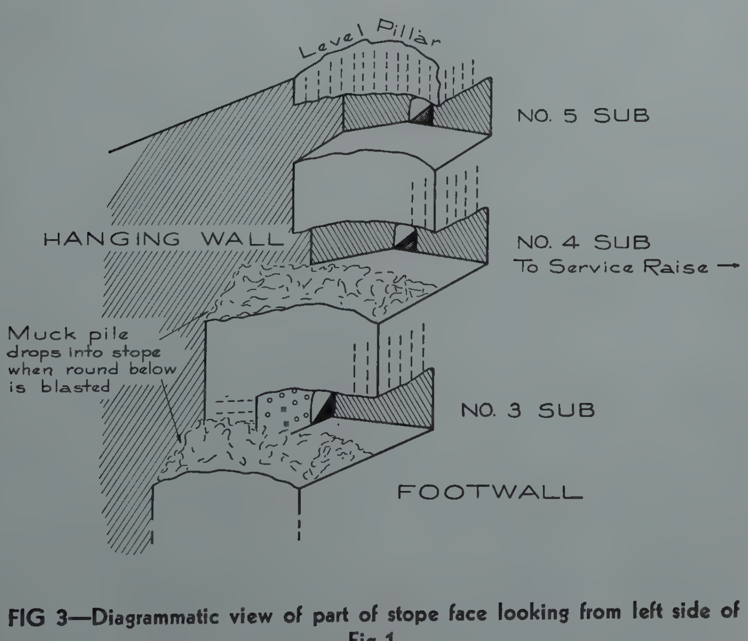

FIG 3—Diagrammatic view of part of stope face looking from left side of Fig 1.

Summary of Advantages

The advantages of sublevel stoping can be summarized about as follows:

Safety factor is high. All work is

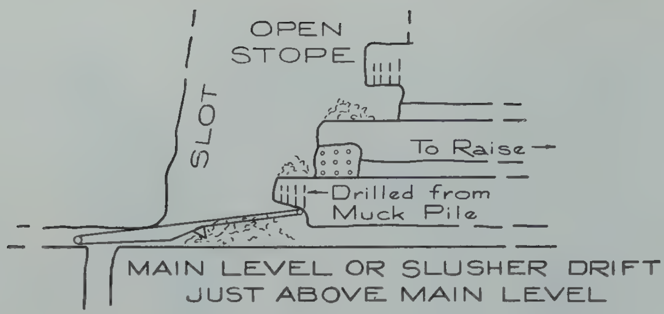

FIG 5—Stope layout without finger raises.

done under a close back, easy to inspect and to bar down. Bench backs have to stand for a short time only. Access to working places is direct and is always along established travelways. Ventilation is good. Safety ropes or guard rails should be used when men are working near the open side of benches.

The direct and easy access to working places for men, supplies and equipment results in a minimum of lost working time.

Drilling can be done simultaneously on several benches and relatively large tonnages can be broken and pulled from small stope blocks.

Supervision and supply service are simple and direct.

All stope drilling consists of slab rounds breaking to a free face, using stoper drills. This type of round can be drilled satisfactorily by relatively unskilled miners and gives low explosives cost with close control of primary fragmentation.

The ore is blasted into an empty open stope so that gravity, to some extent, helps the fragmentation. Broken ore rolls down the footwall so that the method may be used when dips are flat enough to cause considerable difficulty in drawing shrinkage stopes. In shrinkage stopes many ores chimney or hang up on the footwall even with steep dips.

Ore is pulled as fast as it is broken. There is no build up of broken ore inventories that tie up working capital. This also eliminates post breakage oxidation of sulphide milling ore with its frequently adverse effect on mill recoveries. The method does, however, provide for underground storage and continuous stoping during periods when hoisting is interrupted or during milling plant shutdowns.

In deposits with irregular or assay walls there is opportunity to check the walls at each bench from each sublevel. Faces are always accessible and easily sampled. Variations in ore width are readily noted and mining widths can be adjusted during stoping. There is assurance that any ore making out in the walls will not be overlooked and such ore can be mined without difficulty. Waste or low grade areas can be recognized and left on the walls or as pillars.

Full scale production is possible shortly after the block is opened up. Work on haulage levels and draw points can be carried on concurrently with production after the first draw points in the slot area are ready for use.

A moderate amount of slabbing from the walls is not serious since the stope benches retreat rapidly and such slabbing will occur usually over abandoned draw points.

These factors combine to give high tonnages per man shift with low supply, service, and supervisory costs. Clean mining, high underground extraction, and maximum metallurgical recoveries help the overall profit.

Disadvantages

Sublevel mining has the disadvantages common to any open stope method. Main levels and some raises in the larger ore bodies must be protected by pillars, with plans made for pillar recovery after the initial stoping. Probable caving and its results must be considered as pillar robbing proceeds and unsupported wall spans become greater. In the smaller ore bodies these problems are less important.

Some knowledge of the general extent and attitude of the ore body is desirable in planning the mining layout. Development raises and intermediate levels driven while prospecting can usually be fitted into the final plans.

The tonnage from the sublevel drifts is extracted at a relatively high cost. However, even with narrow ore widths this high cost applies to only a small percentage of the total stope tonnage.

Irregular breakage on the benches may leave some ore that is difficult to reach with standard equipment. We found that two or three sets of jointed rods for holes up to 25 ft would take care of these situations without difficulty.

Examples

At Copper Canyon the ore body was continuous over a considerable length, varied from 15 to 80 ft in width with a dip of about 70°. Individual stopes were about 150 ft high and up to about 100 ft in length. Over a period of 20 months the average tons broken per machine shift in sublevel stopes was 105 and average per man shift was 42.7 tons. Pounds of powder per ton broken was 0.74.

At Darwin small ore bodies of widely different shapes have been successfully mined with sublevel stopes. One ore body had a pitch of about 40° and a vertical extent of 90 ft. The cross section was a rough triangle with base and altitude about 100 ft. A slusher layout was used to clean up along the flat footwall trough. Another ore body was a nearly vertical lense, 60 ft long, 80 ft high, and from 10 to 30 ft wide.

Direct costs on these stopes were, respectively, $0.50 and $0.76 per ton lower than the average of other open and shrinkage stopes during the same period. Some open slusher stopes on flat, moderately thick bedded ore gave costs that compare favorably with the sublevel costs.

Summary

Where open stopes are feasible and the vertical extent of the ore body is appreciable, sublevel stoping gives clean, low cost mining, with a high degree of safety, operating flexibility, and assay control.

Acknowledgment

For many of the operating details and data discussed here the writer is indebted to Mr. L. E. Snow, Mr. S. K. Droubay, and their operating staffs.

Jaw Crusher Capacities (Blake Type)

By D. H. GIESKIENG,* Member AIME

Published tables of jaw crusher capacities are compiled for the most part from field performance data, interspersed with interpolations, extrapolations, various safety factors, and other modifications. Such a table attempts to balance and equalize field data from widely different materials, with differences of crushability, efficiency of feeding, sizing of the feed, and so on.

The tests described in this paper were undertaken with the objectives of checking previously reported capacities; of studying the effects of various factors upon the capacity, including jaw plate curvature; and, if possible, of deriving a general capacity equation. Such an equation has been derived from the data obtained in these tests. It is given together with factors for its use. While more accuracy than is usually required may be obtained by careful application of the equation, it is of simple form and provides a convenient reference plane upon which the potential capacity of various crushers may be compared, and with which operating adjustments may be made on existing installations.

The principal deviation in these tests from the usual approach consists of a laboratory technique which has been developed for the production of selected capacity data. Recording instruments are connected to crushers in such a manner that a continuous record is produced which discloses the inherent maximum capacity of the crusher and shows capacity decreases caused by hangups. The capacity effects of various crushing conditions are made more evident by this arrangement.

The recording instruments were first applied to a standard sized 10 by 7 in. Blake-type jaw crusher in the laboratory. Sufficient tests were run on this with various combinations of setting, throw, speed, feed sizing, shapes of jaw plates, and other operating features, to establish the individual effect of each on the capacity. The instruments later were applied to much larger crushers in the field, through the excellent cooperation of the managements in the Mesabi Iron Range, and data were obtained which closely corroborated the established trends.

San Francisco Meeting, February 1949.

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* Formerly Crushing Cement and Mining Machinery Research Section, Allis-Chalmers Manufacturing Co., now Sales Representative, Denver District Office.

The capacity equation resulting from the test work is made up of a series of factors, each representing the contributing effects of setting, speed, width of chamber, nip angle, feed density, type of jaw plates, stroke, and the overall operating technique. Fortunately, most of these factors were revealed to be in direct proportion to the capacity for straight plate chambers. This equation is also applicable to chambers using curved plates, with slight modifications resulting from the capacity limitation of the upper portion of the crushing chamber.

The equation has been derived from tests covering a very wide range of feed density and hardness. It has been found valid, providing that the feed material is either nonpacking in character, as is usual, or allowance in designing the crushing chamber shape has been made to accommodate this characteristic.

Preliminary investigation has shown that the equation is also applicable to single-toggle jaw crushers. Conditions of this are that the top of the flywheel rotate towards the top of the crusher opening, and that the horizontal motion of the swing jaw relative to the stationary jaw at the discharge opening be considered as the stroke. The relatively large upper chamber jaw motion would probably result in realiza-

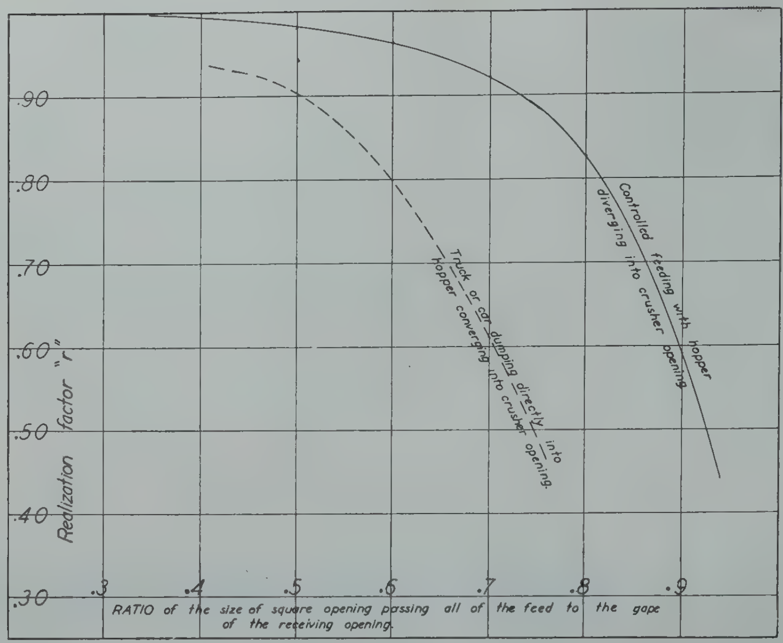

FIG 1—Approximate realization factors for use in the capacity equation.

tion factors slightly higher than those given in Fig 1, inasmuch as minor feed bridging problems are less probable.

The Equation

The results of the laboratory and field tests are summarized in the following equation, which can be applied directly where the crushing plates are straight, and either corrugated or smooth:

$$C = f \cdot d \cdot w \cdot y \cdot t \cdot n \cdot a \cdot r \quad [1]$$

where

C is the capacity in short tons per hour through the crusher.

f is the feed factor, dependent upon the presence of fines in the feed, and the surface character of the jaw plates used.

Values of "f"

	Smooth Plates	Corrugated Plates
With normal fines....	0.0000414	0.0000305
Fines scalped out.....	0.0000364	0.0000254
Large pieces only.....	0.0000312	0.0000207

d is the apparent density of the broken product in pounds per cubic foot. (If the true specific gravity is known, 40 pct voids may be assumed and d becomes 37.4 times the specific gravity of the ore.) This is shown in Fig 2.

w is the width of the crushing chamber in inches.

y is the open side setting of the crusher in inches. (This is measured from the lower tip of the swing jaw, and in the case of corrugated plates from the tip of the corrugations on the swing jaw to the opposite valley on the stationary jaw plate.)

t is the length of stroke in inches at the lower tip of the swing jaw plate.

n is the crusher revolutions per minute or crushing strokes per minute.

a is the nip angle factor. It is unity for 26° and 3 pct greater for each less nip angle degree. For example, a nip angle of 20° has an a value of 1.18.

r is the realization factor. It is unity for perfectly uniform choke feeding and usually less for actual operating conditions; according to the feeding method used, and the probabilities of hangups involving the size of feed and the crusher opening. Approximate values are given by the curves in Fig 1. These values are further reduced by intermittent feeding.

Curved Jaw Plate Capacities

If the crushing plates are curved, either corrugated or smooth, the factors given above are the same, with the provision that the nip angle is taken in the portion of the upper chamber which is usually flat. If a continuous jaw plate curvature is employed, the nip angle should be taken in the crushing chamber where the largest pieces in the new feed would lodge at their minimum (slot size) dimension.

An alternate, more complicated method of predicting curved plate capacities is outlined in the text. Accuracy comparisons between the two methods are shown graphically in Fig 8.

Test Work

LABORATORY TESTS

The 10 by 7 in. Blake crusher tests were recorded by means of a 35 mm motion picture camera. This was driven by the crusher through a flexible shaft, thereby synchronizing exposures with the crushing cycle. The camera photographed an instrument panel which contained devices to indicate the crusher speed, cumulative product weight, motor power conditions, the number of the crusher revolutions, and a card identifying the test. Two pictures were taken per revolution of the crusher flywheel. Following several tests, the film was developed and read in a microfilm viewer as a negative, the data being plotted as functions of crusher revolutions.

In plotting the data from the film it was found to be advantageous to have each frame identified by a number so that quick reference could be made to any given period in the test. Accordingly, a revolution counter driven by another flexible cable was placed in view of the camera.

The crusher speed was brought to the instrument panel by means of a

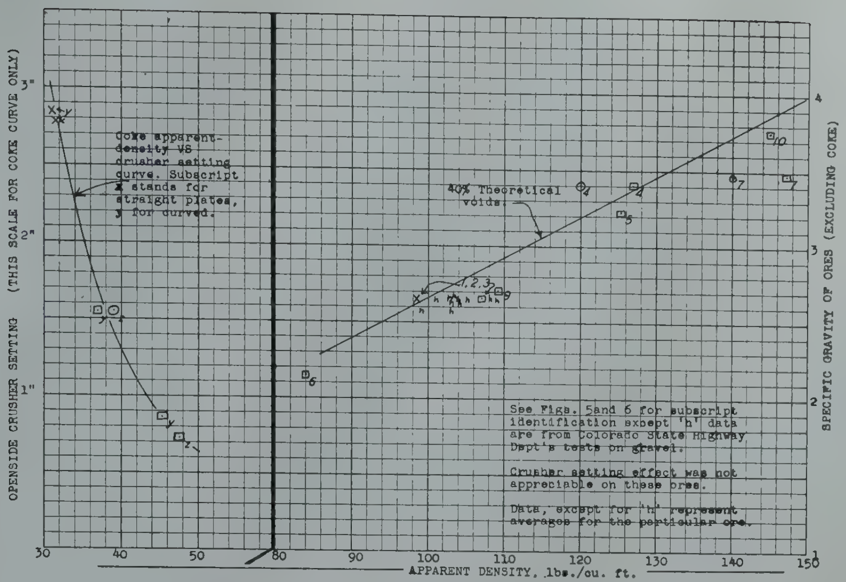

FIG 2—Apparent density of crusher products.

remote indicating tachometer of the generator-voltmeter type. This was very useful in determining the effect of speed on capacity during tests where the speed was varied purposely.

The power requirements of the crusher motor were indicated by a wattmeter. The readings shown by this instrument were plotted along with the capacity curves and aided considerably in their interpretation. These power curves have been integrated with a planimeter to determine the actual kilowatt hours per ton required for crushing.

Several means of continuously weighing the crusher product were tried. The method finally adopted consisted of catching the product in a box on an air-inflated automobile inner tube and interpreting the weight as a function of the increasing pressure of this air. The inner tube was inflated initially to a pressure of about 10 in. of water. The valve stem was connected by a rubber hose to a manometer filled with a colored liquid. The U-shaped manometer tube served to seal the inflating air in the inner tube, as well as to indicate the weight of the crusher product in terms of liquid level.

The particular 6.00 by 16 inner tube used was found to give a uniform weight-pressure response up to loads of at least 150 lb. An arbitrary linear scale was placed alongside the manometer to graduate the column rise.

It was found to be advantageous to determine the rise and equilibrium conditions of the manometer from the data plot, and to recalibrate the weight values of the manometer graduations for each test by weighing the sample after each run. There were slight changes of calibration from run to run, caused either by stretching of the rubber, slight thermodynamic changes of the inflating air, or by disturbance of the apparatus while handling the sample. These changes are compensatory during the test itself but preclude a fixed calibration of the manometer.

About 50 lb of feed were sufficient for most of the 10 by 7 in. crusher tests, and these were recorded on approximately 15 ft of 35 mm film. This amount of film permitted the camera to be started just before the feed reached the crusher, and allowed it to run until crushing was completed and equilibrium conditions were again apparent to the operator.

FIELD TESTS

Where a weightometer or other similar conveyor weighing device is installed in conjunction with a crusher it often may be used to give a continuous record of the weight output, either by photographing the totalizer against time, or by some chart scribing means connected to the weighing beam. In one instance, the settling rate of a large railroad car on its coil springs was photographed against time as it received the product from a 60 by 48 in. crusher directly overhead; the relationship was found to be sufficiently linear for interpretation. These methods and field applications of the inner tube method were used to obtain data to check the extrapolation of trends established in the laboratory work.

In preparation for a test the crusher bearings were allowed to warm up for at least a half hour by running idle. The samples were choke-fed by hand to the smaller crushers, while the larger crushers were choke-fed by collecting a sample on the apron feeder with which each was equipped.

Test Interpretations

While several of the factors influencing crusher capacity have been established by various investigators, several more have been considered as still open questions.[1] Since one of the prime objectives of this investigation concerned the influence of jaw plate curvature on capacity, it was deemed expedient to review the entire situation in as much detail as possible, so that effects caused by curvature could be evaluated as such. Fortunately, most of the variables were either confirmed or found to be directly proportional to the capacity. The few deviations from this were not serious, and proportionate assumptions appear to be adequate for all practical purposes.

SPEED

Speed was one of the first effects to be tested. This was done by driving the crusher with a variable-speed direct current motor and varying the crusher speed during the test run. The speed range extended from about 40 pct above normal to about half speed.

[1] References are at the end of the paper.

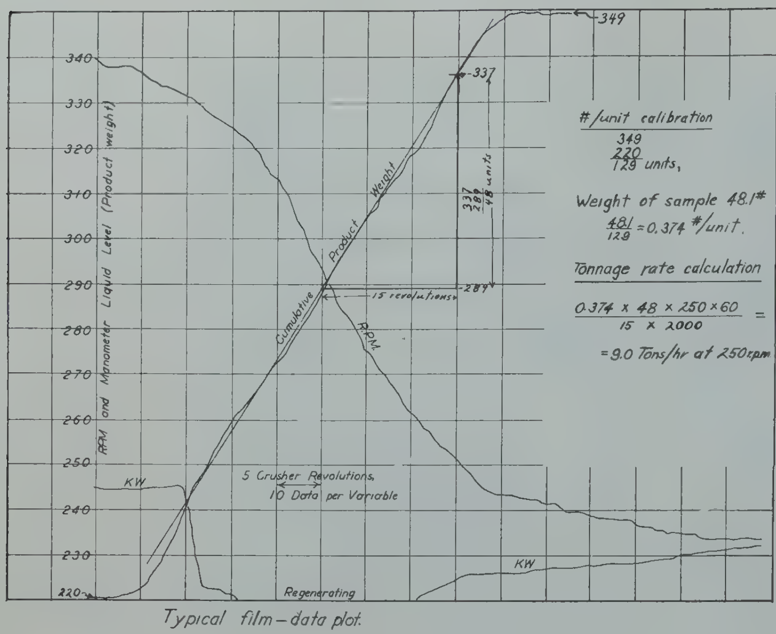

FIG 3—Typical film-data plot.

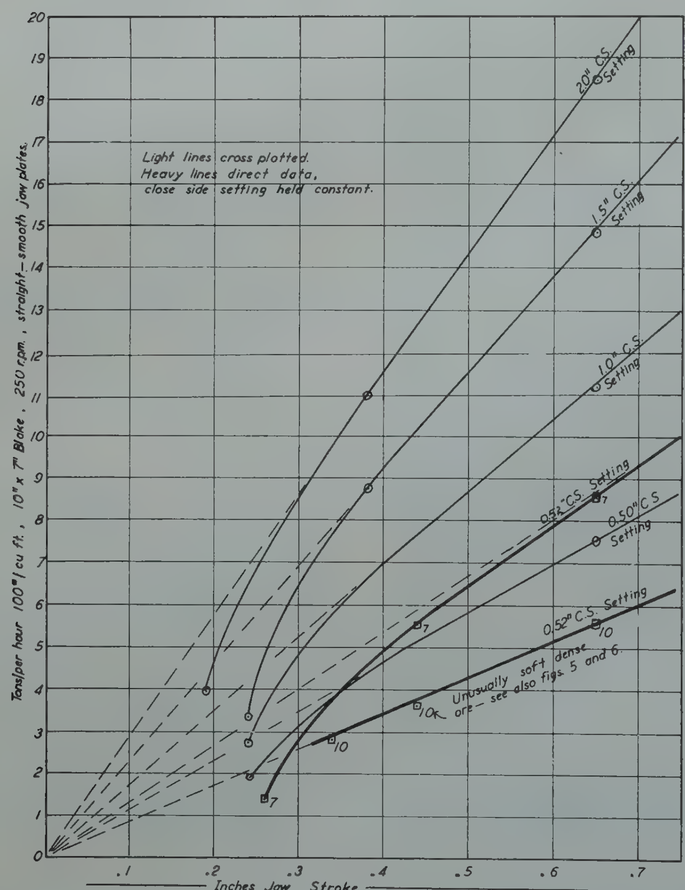

FIG 4—Jaw stroke effect upon capacity.

As previously outlined, the camera was synchronized with the crushing cycle, and when the cumulative weight output was plotted it became a practical indication of the output per revolution. These data were found to be substantially constant for several materials and plate curvatures. It follows that for normal jaw crusher speeds, capacity is proportional to speed. This agrees with Hersam's formula published in Taggart,[1] and is usually accepted in the field.

A typical plot of a variable speed test is shown in Fig 3. It is noted that when the power to the crusher was reduced, the amount of product per revolution remained constant as the crusher speed decreased from 335 to 254 rpm. The crusher capacity was 9.0 tph at 250 rpm.

The cumulative weight curve shown is similar to that of many of the constant speed tests, particularly where fines were included in the feed.

WIDTH OF CRUSHING CHAMBER

Capacity is usually considered to be proportional to the width of the crushing chamber. While certain analogies from hydraulics would seem to indicate a slightly greater than proportional increase, this deviation is apparently negligible, since a proportional assumption resulted in successful reduction of 60 by 48 in. and 26 by 19 in. crusher data to equivalent 10 by 7 in. data (Fig 5). Capacity is, therefore, considered to be proportional to the width of the crushing chamber.

JAW STROKE

The length of the jaw stroke in the 10 by 7 in. Blake laboratory crusher was varied by raising and lowering the rear toggle seat. The resulting effect of stroke on capacity was analyzed by cross plotting data corrected to common conditions of speed, nip angle, and settings, and by running a series of tests where the close-side setting was held constant and the stroke varied.

A typical cross plot, that of the straight-smooth plates, and also direct data, are given in Fig 4. It is evident that above a practical minimum, ca pacity is directly proportional to stroke.

SETTING

All of the laboratory capacity data obtained with normal jaw strokes were corrected for minute differences of stroke, characteristic of the toggle system at various settings, to a mean of 0.65 in. They were also corrected for product density to 100 lb per cu ft and for differences in speed to a common base of 250 rpm.

These data so corrected were plotted as a function of open-side setting and capacity. Fig 5 and 6 concern straight plates and when extensions are made of similar feed conditions it is evident

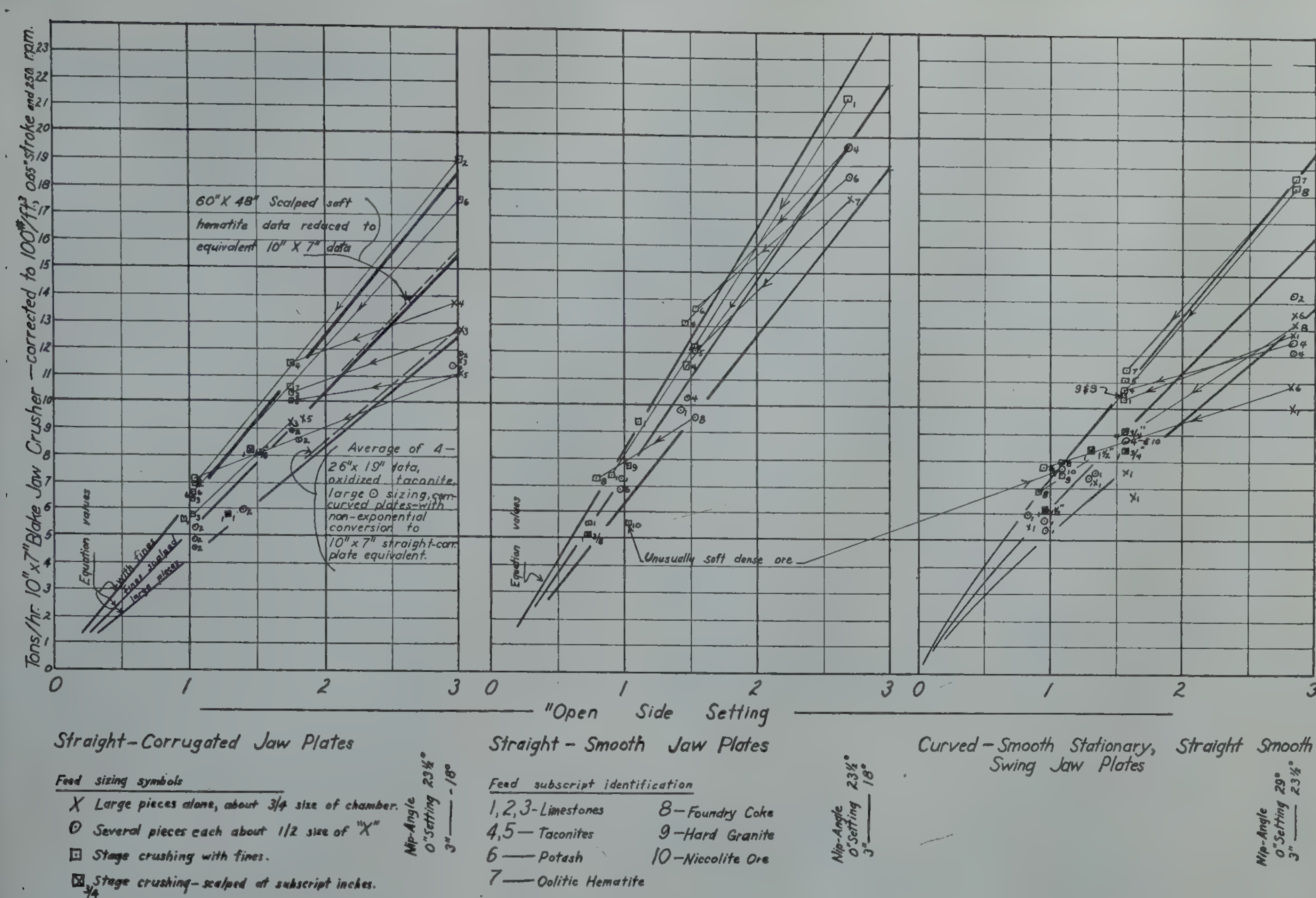

FIG 5 FIG 6 FIG 7

FIG 5–7—Summary of laboratory and converted field data plotted as open-side setting vs. capacity.

that these pass through the origin, and hence, *straight plates have a proportionate setting-capacity function.*

Fig 7 is the summary plot of laboratory data obtained with smooth-curved plates. These plates were constructed with a full exponential curvature in the lower zone to emphasize the curved plate characteristics which are generally regarded as desirable. Setting for setting, these curved plates had the same gape and the same depth of crushing chamber as the straight plates, but the nip angle of the upper chamber was 5½° greater. (For most applications this amount of curvature is not required or used, but by analysis we can predict with fair accuracy what the performance of plates having a lesser degree of curvature would be.)

The setting-capacity curves on Fig 7 are apparently exponential, and an attempt follows to predict the effect of plate curvature on this basis.

Fig 8 is a composite graph showing the "with fines" capacities of straight-smooth plates and also curved-smooth plates. The actual data are shown in heavy lines while the synthesized data are in light lines. In this graph it may be seen that beyond a setting of about 1 in. the curved plates have a tested capacity less than that for the straight plates. It may be reasoned that if a succession of jaw plates are considered having less and less curvature but the same gape and setting, they would begin to have the same nip angle in the upper chamber and also the same capacity characteristics as the straight plates. Accordingly, the straight plate capacity may be used as a maximum reference and the curved plate capacity approximated by a skew function.

In computing a factor for the skew analysis it may be seen that to conform, it can be of exponential type and fortunately of relatively small degree variation. Accordingly, an exponential consideration is made of the open-side setting with the degree controlled by the percentage of nip-angle change.

The corrective factor just discussed becomes:

$$\frac{y^x}{y} \qquad [2]$$

where y is the open-side setting and x the decimal expression of the percentage of nip-angle difference between the straight plate and curved plate crushing chambers of equal setting and gape.

As an example of the application of this factor, the calculation of the capacity of the curved plates tested in the 10 by 7 in. crusher is made at an open-side setting of 3 in., using the straight-smooth plate data obtained at 3 in.

$$y = 3$$
$$x = 18/23.5 = 0.766$$
$$\frac{3^{0.766}}{3} = \frac{2.34}{3} = 0.78$$

The straight plate capacity at 3 in. is 25.0 tph. The curved plate capacity becomes 25.0 times 0.78 or 19.5 tph. By plotting 19.5 tph on Fig 8, at the 3 in. setting and other points, it is evident that the accuracy of the method is sufficient for most settings. The X's on Fig 8 represent skew points that have been calculated.

This method of skew computation requires that the straight plate dimensions of the crusher be known before

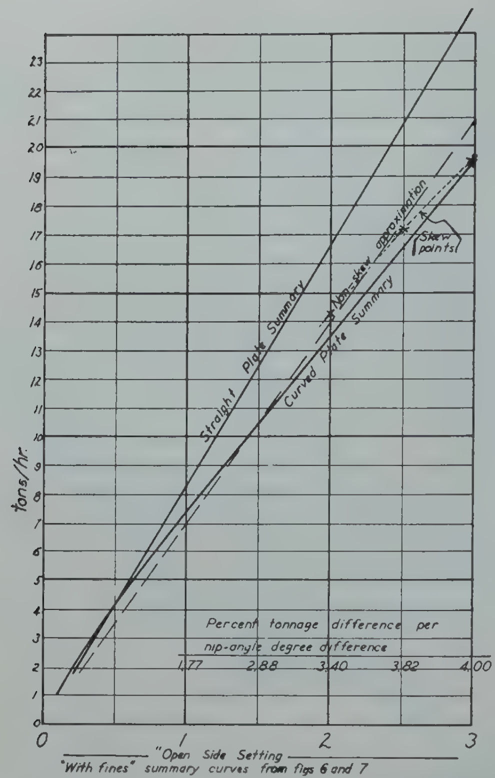

FIG 8—Curved jaw plate capacities calculated by exponential skew and simplified linear methods compared with data summary.

the straight plate capacity potential can be calculated and converted to curved plate equivalents by factor (2).

The foregoing method may not be particularly convenient in the field, and it is suggested that practical results may be obtained by merely applying the equation as listed at the discharge of the curved plate chamber but taking the nip angle from the upper portion of the crushing chamber. Using this latter approximation the straight-smooth plate data are converted on the basis of 3 pct per degree nip angle and are included on Fig 8. Here it may be seen that this more convenient method has relatively high useful accuracy.

Where a continuous plate curvature is employed from the top to the bottom of the jaw plates it is suggested that the nip angle be taken at a point in the chamber where the opening is small enough to lodge the largest pieces of the feed, since the breaking capacity of the upper chamber appears from the foregoing to have an almost integral effect on the overall crusher capacity.

Nip Angle

The nip angle effect was tested directly by blocking straight plates at various angles in the 10 by 7 in. crusher. While these tests had evidence of high accuracy they are relatively few in number, and reference is made to Fig 8 in order to obtain what corroboration may be had.

General consideration of Fig 6 and 7 indicates that the breaking capacity of the curved plates used in Fig 7 is not sufficient to supply the increased capacity capabilities of the discharge section, which has a much more capacity-conducive nip angle. Accordingly, the curved plate chamber becomes to all intents and purposes a straight plate chamber, at least in the larger settings, with the provision that the effective nip angle in this case is about $5\frac{1}{2}°$ larger. Because of this the difference in the two direct data curves on Fig 8 can be evaluated in terms of percentage of capacity per degree of nip angle difference. These values have been calculated for several points and are included in Fig 8 above the setting to which they apply. It is evident that these figures approach a limit a little over 4 pct per degree, which tends to confirm approximately the 3 pct per degree value found in the direct nip angle tests from 18 to 24°.

Hersam,[2] working principally with a Dodge type jaw crusher, theoretically deduced that capacity was proportional to the cotangent of the angle of nip, $\left(\frac{d}{e-b}\right)$. This was corrected in part by a variable equation factor k which was evaluated by experiment.

The cotangent theory directly and the limit shown in Fig 8 to a lesser extent indicate that the nip angle effect per degree will increase as the nip angle is decreased. However, jaw crushers are limited to a rather small nip angle range, and since a constant assumption simplifies the equation and tends to compensate losses incident to higher capacity rates, as did k, the nip angle effect is considered to be a constant 3 pct per degree. The f factors given with the equation are based upon this figure, and as outlined with the equation a nip angle of 26° is the reference angle and each degree less increases the capacity 3 pct. The field data shown on Fig 5 were converted on this basis through a range of 7 and 8°.

Crushability

As noted on Fig 5, 6, and 7, the data plotted have been corrected to common conditions of speed (250 rpm), stroke (0.65 in.), and apparent density of the product (100 lb per cu ft), but not for an equally wide variety of hardnesses and ore strengths. The lack of a "crushability-capacity" trend is obvious within a relatively small band of experimental error. (Particular reference is made to the points representing feeds with fines.) Since a considerable number of combinations and wide ranges of hardness and density were tried in the tests it seems reasonable to disregard the effect of feed strength or hardness on capacity.

Feed Factors

Tests made with feeds containing the fines resulted in smoother operation and more continuous product weight curves plotted from the film data. These tests were quite reproducible as evidenced on Fig 5, 6, and 7. Feeds without fines often had minor interruptions which necessitated that an average be taken on the plotted cumulative weight curve of the test. While the nature of these interruptions was portrayed by the shape of these curves, which greatly assisted in determining a mean, these averages were not too definite and resulted in a fairly wide

spread of points on Fig 5, 6, and 7. Accordingly, these points in turn are averaged to determine the feed factors pertaining to feeds without fines. Fortunately, the field data converted on Fig 5 corroborates these feed factors, which take a fairly uniform spread through Fig 5, 6, and 7.

Density

Most of the products from the tests were weighed in a container of known volume, and the apparent density, or pounds per cubic foot was determined. These values were carefully averaged for each type of ore and the resultant was used to correct each of these ore varieties shown on Fig 5, 6, and 7, where even the changing density of coke due to crusher setting was compensated. The relatively large number of apparent density determinations made in these tests assisted the evaluations, since individual determinations are often erratic.

Specific gravity determinations are usually more consistent and convenient for individual tests, and the large majority of broken ores have been found to have about 40 pct voids. Accordingly, substitution may be made of the apparent density factor in the equation for most ores by the following equivalent:

$$d = (62.4 \times 0.60 \times \text{sp gr of ore}) = (37.4 \times \text{sp gr of ore}) \quad [3]$$

While this substitution may be more convenient, experience has shown that there are some exceptions. Fig 2 is included for convenience on this topic.

Realization Factors

Assuming that the angle of nip is adequate, and that the feed is of the usual type and does not tend to pack excessively, the equation, exclusive of the term r represents the maximum potential capacity of the crusher. In actual practice this figure will seldom be obtained, but will be reduced by interruptions incident to hangups and other conditions resulting in intermittent feeding. Curves giving approximate realization factors for several conditions of feeding have been included in Fig 1.

Two unusually complete sets of crushing data are given in the literature by Sheppard.[3] These are substituted in Eq 1 which is then solved for realization factors characteristic of the installations.

CRUSHER NO. 1

48 by 36 in. Blake type jaw crusher
Jaw plates—straight corrugated
Feed—minus 24 in. (square opening) limestone with fines
Apparent density—106 lb per cu ft
Feeder—carloads dumped in converging hopper
Width—48 in. Swing—1½ in.
Setting o.s. 8⅜ in. Speed—155 rpm
c.s. 6⅞ in. Nip Angle—26°
Capacity—4½ ton carload in average of 85 sec = 191 tph

Maximum potential capacity by equation:

$$C = f \cdot d \cdot w \cdot y \cdot t \cdot n \cdot a = 0.0000305 \times 106 \times 48 \times 8.375 \times 1.50 \times 155 \times 1.00 = 302 \text{ tph.}$$

Realization factor, $r = 191/302 = 0.632$ or 63.2 pct.

CRUSHER NO. 2

60 by 48 in. Blake type jaw crusher
Jaw plates—straight corrugated
Feed—minus 34 in. (square opening) fine grained gray biotite granite with fines
Apparent density—108 lb per cu ft
Feeder—skip loads discharged directly into crusher
Width—60 in. Swing—1¼ in.
Setting o.s. 7½ in. Speed—167 rpm
c.s. 6¼ in. Nip angle—24½°
Capacity—11,615 lb in 70 sec = 299 tph

Maximum calculated capacity:

$$C = f \cdot d \cdot w \cdot y \cdot t \cdot n \cdot a = 0.0000305 \times 108 \times 60 \times 7.5 \times 1.25 \times 167 \times 1.045 = 323 \text{ tph}$$

Realization factor $= r = 299/323 = 0.925$ or 92.5 pct.

The 48 by 36 in. crusher was fed by a car dumping sideways onto the slope of a hopper converging into the receiving opening.[a] Since the car was actually longer than the crusher opening width, the descending feed was crowded and subjected to hangups and interruptions in its entry into the crushing chamber.

The 60 by 48 in. crusher was fed by a skip which discharged directly into the receiving opening.[a] The skip opening was smaller than that of the crusher and the skip itself served to shape the feed ideally for entry into the crusher. Even though the size of feed to size of crusher opening ratio is about the same in either of these cases, the more favorable feeding conditions pertaining to the 60 by 48 in. crusher were apparently responsible for an increase of about 30 points in the realization factor.

The above calculations consider only the actual feeding period, and additional allowance must be made for time lost between the carloads or skiploads.

[a] Private communication.

Conversion of Field Data to 10 by 7 in. Equivalent

A 60 by 48 in. Blake jaw crusher with straight corrugated plates was tested in the field by receiving the product directly in a large railroad car and measuring the settling rate of the car body on its coil springs. A multiplying lever device was placed centrally under the bridge beam of the car; its movements resulting from the settling of the car, together with an electric clock, were photographed by a movie camera. The resulting data were found to be sufficiently linear and continuous to disclose the inherent maximum capacity of the crusher at the tested conditions.

This crusher was found to have a measured uninterrupted maximum capacity of 466 short tons per hour, while crushing scalped hematite weighing about 155 lb per cu ft. The width of the chamber was 60 in., the open side setting 8.0 in., the stroke 1.44 in., the speed 163 rpm, and the nip angle 25°.

In converting this capacity to 10 by 7 in. maximum equivalents we consider the following conditions in the 10 by 7 in. crusher:

100 lb per cu ft material
10 in. crushing chamber width
3 in. open side setting
0.65 in. stroke
250 rpm
a nip angle of 18° at this setting.

(Both crushers are equipped with straight corrugated jaw plates.) The equivalent capacity of the 10 by 7 in. crushers is calculated as follows:

$$C_{10 \times 7} = 466 \times 100/155 \times 10/60 \times 3/8 \times 0.65/1.44 \times 250/163 \times 1.21 = 15.7 \text{ tph.}$$

Because of the linear character of the factors, this point may be joined by a straight line with the origin as shown on Fig 5 to determine approximately equivalent capacities at settings of less than 3 in.

It should be noted that Sheppard's

tonnage data were average operating performances exclusive of the loss of time between car or skip loads, and included realization factors characteristic of the installations. The tonnage figure of 466 obtained with this crusher was the maximum inherent capacity. The average capacity for the whole test indicates an experience factor of about 95 pct, which is consistent with the size of feed and feeding conditions, that is, feed size to crusher gape ratio of about 0.5 to 0.7, apron feeder, and nonconverging hopper.

The 26 by 19 in. crusher data shown on Fig 5 were obtained by a field application of the inner tube method. They were converted on a factor basis similar to the above calculation. The nip angle of the upper chamber was 26° and the product weighed about 122 lb per cu ft. Tests were made at open side settings of 3¼ and 2½ in. The converted data of different test runs were 12.7, 14.3, 11.4 and 13.0 tph at the 10 by 7 in. crusher setting of 3 in. The plot on Fig 5 is an average of these converted tonnages.

Other Characteristics of Crushing Chambers

While this paper is principally concerned with the tonnage rates of various Blake type crushing chambers it would seem to be worth-while to deviate at this point and go briefly into some of the other characteristics.

CURVED JAW PLATES AND PACKING RELIEF

Apparently most ores have physical properties which permit crushing with straight jaw plates at practical settings and strokes without compaction. A few seem to have difficulty in this respect. Particular attention is called to ore No. 10 and its plotted position in Fig 4, 6, and 7. This ore was unusually dense, relatively soft, and had minute interlacing, slightly malleable veinlets. When tested with straight plates, pancaking, loss of expected capacity and excessive power consumption occurred.

When this ore was tried with the curved jaw plates at the same setting and stroke no difficulty was experienced, the power consumption and capacity being normal.

Similar experience has been encountered in the field,[a] and it is accordingly believed that the use of properly curved jaw plates tends to alleviate packing in these unusual cases.

[a] Private communication.

The design of optimum curved jaw plates requires considerable planning since the upper chamber nip angle may be excessively increased, resulting in loss of capacity and receiving opening without commensurate benefits. Most of the modified jaw plates now offered utilize curvature at one or both ends and a flat surface at the upper or center portion. Both ends are sometimes curved to permit reversibility, to prolong the wearing life with some loss in the effective receiving opening.

It may be stated briefly from the unpresented power and screen analysis data obtained in the tests, that the curved jaw plates could be set up closer than straight plates for a given motor load. The largest particles produced at equal settings of straight and curved plates have about the same size and shape; however, the fines produced with curved plates have a larger average particle size.

In some instances greater wearing life has been observed for curved jaw plates, which may result from distributing the zone of intensive wear over a larger area. In drawing strict comparisons between straight and curved jaw plates it should be noted from the preceding paragraph that straight jaw plates do more crushing per ton of product at a given setting than curved jaw plates.

CORRUGATED JAW PLATES

Corrugated jaw plates, while characteristically having less capacity, produce a considerably more cubical product which may bring a premium price or be otherwise more desirable. They tend to make a finer product than smooth plates at the same setting, with a corresponding decrease in capacity. Abrasive feed may cause rapid wear of the corrugations.

UNUSUALLY DEEP CRUSHING CHAMBERS

While unusually deep crushing chambers have the advantage of a smaller nip angle, this type of chamber also may be seen to have setting-stroke ratios at various chamber levels which are especially favorable for the use of properly curved jaw plates. Also, because of this additional depth, curvature may be developed in the lower chamber without appreciable sacrifice of upper chamber nip angle.

The capacity equation as given is applicable to these unusually deep chambers. It should be noted that, in addition to the capacity increase due to the nip angle factor, the operating realization factor would probably be slightly higher than usual because of the more favorable upper chamber receiving proportions.

Summary

The choice between types of jaw plates for any specific crushing application may be influenced by some of the following factors:

1. Abrasiveness of the feed material.
2. Fines distribution desired in the product.
3. Mineral unlocking at coarse sizes.
4. Cubical product desired.
5. Maximum receiving opening desired.
6. Tendency of the feed to pack in the crusher.
7. Nipping angle required (slippery feeds).
8. Capacity desired.

The overall problem of jaw crushers is seen to be involved, but in some cases there are worthwhile economies to be had through selection of proper crushing plates. It is evident that application should be guided by tests or experienced judgment.

Acknowledgment

In conclusion, the writer would like to express appreciation for the fine cooperation of his associates in the company and the operating companies in the field, who from the first appreciated the possibilities of the investigation and gave every assistance toward its completion. Particular acknowledgment should be given to Fred C. Bond and Fred R. Gruner who also contributed invaluable guidance in the preparation of this paper.

The opinions expressed on the controversial subject of crushing are those of the writer, and do not necessarily coincide in all particulars with those of other members of the Allis-Chalmers staff.

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Pretreatment of Mineral Surfaces for Froth Flotation

By S. A. FALCONER,* Member AIME

Introduction

Much attention and publicity has been given, during recent years, to grinding, classification, flotation, and thickening. The various technical papers, and symposiums held to discuss these important phases of milling, have contributed to a better understanding of the fundamental principles and operating variables involved. Another important part of milling—to the flotation operator—is a treatment step to which frequently too little attention is given—"conditioning," or the treatment the ground mineral pulp receives before it enters the flotation machines.

"Conditioning" is a hackneyed and frequently misused term, commonly employed to describe the pretreatment of surfaces of minerals in ore pulps or mill products prior to froth flotation. In general, such pretreatment is for the purpose of changing the flotation characteristics of these mineral surfaces to enable them to respond in some desired predetermined manner when subjected to flotation. Taggart[1] states that the function of conditioning is "either to insure selective collector coating of one mineral species while another species is not; or to take such steps as will compensate for departure therefrom."

As O. C. Ralston has stated:[2]

> . . . we are still faced with the fact that some mineral particles are water-avid, others air-avid, and that by properly modifying them they may be enhanced or diminished in desirable or undesirable avidity. Coatings on minerals must still be assumed to fall into three classes:
>
> (1) a coating of immiscible oil,
> (2) a layer of adsorbed substance,
> (3) a chemically produced coating.

El Paso Meeting, October 1948.
TP 2593 B. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Aug. 30, 1949. Manuscript received Nov. 29, 1948.
* Mineral Dressing Engineer, American Cyanamid Co., Stamford, Conn.
[1] References are at the end of the paper.

It is submitted that the process of preparing mineral surfaces for subsequent flotation treatment should take into consideration physical as well as chemical means for altering the normal flotation response of the mineral surfaces. A more descriptive word than "conditioning" is needed to designate broadly these procedures.

The minerals in many ores cannot be made to respond in a normal manner to usual methods of "conditioning" with selectivity modifying reagents, until they have been cleansed, or interfering slimes and soluble salts have been removed from the ore. Therefore, it appears important to include under the general subject of "conditioning" a discussion of the purposes, methods used, and some of the advantages of these pretreatment steps. This is dealt with under the subject heading "Initial Treatment of Mineral Surfaces and Ore Pulps."

Under this broader interpretation, the methods employed for treating mineral surfaces to alter their flotation characteristics may be divided into three general classes:

1. Physical methods.

2. Chemical methods.

3. Combination of physical and chemical methods.

PHYSICAL METHODS

These include:

1. Scrubbing or attrition grinding to remove surface stains or alteration products from mineral surfaces to assist action of chemicals in subsequent pretreatment and flotation steps.

2. Desliming to remove interfering slimes or soluble salts prior to introduction of chemical reagents to assist flotation.

3. A combination of scrubbing followed by desliming prior to subsequent chemical treatment.

4. The mechanical means used to bring reagents in contact with mineral surfaces.

CHEMICAL METHODS

These include:

1. Addition of surface *cleansing* agents such as acids or alkalies to assist the removal of surface stains and alteration products prior to treatment with other reagents whose function is to modify further the normal flotation response of the mineral surface.

2. Corrective agents to prevent formation of slime coatings, or assist dispersion of other reagents.

3. Addition of surface modifying agents to activate or depress the mineral in the subsequent flotation step.

4. Addition of promoters and frothers to float the selected mineral or minerals.

ADVANTAGES OF PRETREATING PULPS BEFORE FLOTATION

These may be listed as follows:

1. Improved metallurgical results due to better contact between mineral surfaces and reagents. Also, apparatus used for pretreatment can provide sufficient capacity to level out minor inequalities in rate of feed and characteristics of the ore.

2. Reduction in reagent consumption.

3. Reduction in time required to complete reactions between reagents and mineral surfaces.

4. Reduction in number of flotation machines.

5. Possibility for more intimate contact and better spreading of reagents which are viscous and poorly soluble in water, for example, treatment of phosphate rock with fatty acid and fuel oil.

Initial Treatment of Mineral Surfaces and Ore Pulps

DESLIMING OR WASHING FOR REMOVAL OF SLIMES, COLLOIDS, AND/OR SOLUBLE SALTS

Most ores contain alteration products either in the form of soluble salts or colloids. If these are present in appreciable amounts, they may interfere with flotation and must be removed. The soluble materials may react chemically with the minerals in the ore and interfere with flotation. The colloidal material and slimes can interfere in a number of ways, namely, (1) by consuming reagents—thus decreasing separating efficiency and increasing the cost of treatment, (2) by forming coatings on other minerals and thus preventing their selective separation by flotation, or (3) by diluting the grade of the concentrate. Flocculated colloids or slimes can also interfere by enclosing fine particles of valuable mineral and thereby preventing recovery of the latter.

Slimes

In the discussion following presentation of a paper[3] given at the February 1930 meeting of the AIME in New York, the opinion was expressed by Prof. A. F. Taggart and Prof. A. W. Fahrenwald that difficulty in floating overground sulphides is not due to fineness *per se* but is from primary slimes which adhere to the sulphide particles and prevent adherence of the mineral to an air bubble. At the time, Professor Taggart observed that in general, when the surfaces of sulphide particles are slime-coated there is poor flotation, and that a reagent that affects and lessens the slime coating generally aids flotation. Commenting further, he described certain experiments with pure galena in distilled water into which was introduced a suspension of ground quartz. The latter did not coat the galena, but when a small amount of lime was added a firmly adherent layer of quartz particles formed on the galena and prevented an air bubble from attaching to the latter. On the other hand, when the clean galena particles were placed in a distilled water suspension of slime from run-of-mine Anaconda ore it was found that a bubble would not adhere to the galena surfaces in the presence of a collecting agent, but the addition of lime freed the surface of slime and permitted flotation.

G. R. M. Del Giudice[4] in a discussion of the effects of slimes on flotation, reports that:

> . . . experiments indicate that the mechanism of slime coatings may be due to a reaction between anchored ions on the mineral surfaces to give a relatively insoluble compound which will then act as the cement between the slime particles and the minerals. The reagents which inhibit slime coatings seem to be those which react with the surfaces of either the mineral to be floated or of the slime particles, to form thereon compounds which are less soluble than the compound formed by the interaction between the soluble collecting agent and the salt or salts present at the mineral surface. Inasmuch as the chemical composition of different ores varies, and the slime-coating inhibitors apparently function by chemical reaction, that these reagents must in general vary with the ore seems clear. Water glass, however, appears universally applicable to sulfide ores on account of the relatively low solubility of metal silicates.

In the treatment of sulphide ores containing interfering slimes, it is usually possible to overcome or minimize their effect by the use of proper controlling reagents. This procedure is not always possible or economical in the case of nonmetallic ores, where the promoter reagents used are normally less selective. Where the slimes virtually inhibit flotation, as in the case of raw Florida phosphate rock, desliming is the accepted practice. Where slimes cannot be discarded, conditioning to render either the slime or the mineral particle too insoluble to permit the cementing reaction discussed in Del Giudice's paper[4] is indicated.

In an interesting paper,[5] S. G. Bankoff postulates that the conditions that control the flocculation of mineral particles in water are the same as those that control slime coating. Evidence is presented to show that slime coating is inhibited whenever the slime particles are ionized sufficiently for Brownian movement to occur and is facilitated when they are not. Both slime coating and flocculation are further aided by water-repellent surfaces on the adhering particles. Adhesion occurs when one surface is coated with a substance that is known to react with the other surface to make it water-repellant or when both surfaces are water-repellant. Gelatin accelerates both slime coating and flocculation tremendously.

Interfering slimes are usually dealt with by desliming in an apparatus such as a hydroseparator, or a bowl classifier.

Recent experiments indicate that the newly-developed Dutch State Mines cyclone thickener may also have application for this purpose.

Various investigators have also proposed the use of a preliminary, selective flotation step to remove interfering slimes from nonmetallic ores, prior to further flotation treatment.

Where separate sand-slime flotation treatment is to be used, it is important in laboratory testing to avoid making tests on samples that have been wetted and then dried or allowed to stand in the form of a pulp for more than a few hours.

An excellent discussion of the advantages of washing and desliming flotation feed is given by A. L. Engel.[6] He cites a number of instances where improved results were obtained by removal of colloidal slimes from the ore. For example, slow elutriation for removal of colloids from a copper ore ground to flotation size resulted in an increase in recovery from 65 to 90 pct. In elutriation, only 1½ g of material was removed from a 1000 g sample. This removed material consisted almost entirely of colloidal silica which had interfered with the flotation of fine particles of bornite.

The extensive studies in connection with the influence of colloids in flotation, conducted by the late W. O. Borcherdt during his association with The New Jersey Zinc Co., should be mentioned. The industry has profited from his valuable work.

Soluble Salts

If present in excessive amount, interfering soluble salts must be removed by washing and filtration, or by countercurrent decantation methods.

It is well known that soluble salts in ores can have a pronounced effect on flotation. Soluble copper salts interfere with selectivity when treating complex copper-lead-zinc sulphides. Zinc sulphate is sometimes troublesome in the treatment of some lead-zinc ores because it interferes with zinc recovery and requires larger amounts of activating agents and promoters. Ferrous, ferric and aluminum sulphates also interfere in flotation of sulphides. In connection with these latter reagents, A. W. Hahn reported[7] that burnt lime eliminates to a large extent the deleterious effect of ferric and aluminum sulphates, but offsets to only a small extent the effect of ferrous sulphates. To eliminate the effect of the latter, he proposed the use of an oxidizing agent such as oxygen, chlorine, or chloride of lime, added to the ore pulp in the presence of burnt lime.

Discussing this subject from another angle, N. H. McKay states[8] that whereas he has noted numerous articles relating to the effect of harmful salts in flotation and the research and mill experiments that have been carried out to obviate their effect, apparently nothing has been published on the *beneficial* effect of the soluble constituents. In this connection he discusses the opportunity of utilizing soluble zinc salts in the treatment of lead-zinc ores to depress zinc, and also to offset the effect of slimes which may protect sphalerite surfaces from surface oxidation. He also cites the apparently beneficial effect of ferrous sulphate in conjunction with cyanide or alkaline sulphites for depressing pyrite during flotation of chalcopyrite in ores mined at the Matahambre Property of the American Metal Co., in Cuba.

CLEANSING OF MINERAL SURFACES BEFORE FURTHER TREATMENT

The beneficial effects to be derived from cleansing of mineral surfaces prior to further treatment with reagents ahead of flotation are set forth fully in an excellent paper[9] by J. E. Norman and O. C. Ralston presented at the February 1939 meeting of the AIME in New York. In this paper the authors express the opinion that:

> . . . experimenters spend too much time seeking new flotation reagents that will give more selective filming, when actually their troubles are due rather to impurities that either mask the true mineral surface or act as activators on surfaces of minerals that are not to be floated.

Using this philosophy as a basis, the authors investigated two general methods of cleaning contaminated surfaces:

1. Chemical preparation,
2. Attrition milling.

These authors define chemical preparation as consisting of washing the flotation feed with solutions of dispersing and/or complexing reagents such as hydrofluoric acid, phosphoric acid, fluosilicic acid, sodium hydroxide, tetrasodium pyrophosphate or sodium cyanide. They report that hydrofluoric acid gave best results in most tests because of its ability to form complexes, with many heavy metal ions and to "loosen" slime.

J. W. Johns, discusses[10] the effects of surface films on metallic gold. The addition of caustic soda (to change the pH to 8.2) or sulphuric acid, apparently removed this film and permitted better recovery. However excess sulphuric acid (above 5 lb per ton) greatly retarded flotation, 74 pct recovery at 5 lb per ton (pulp pH 6.8) to 5.6 pct recovery at 6.0 lb per ton (pulp pH 4.0). On the other hand, caustic soda in amounts up to 4.0 lb per ton (pulp pH 10.4) gave increasingly good recovery—94.5 pct.

Attrition grinding or scrubbing is the recommended mechanical method for cleaning mineral surfaces of slime and stains or deposits of salts. Norman and Ralston developed a special type of scrubber, comprising a high speed, rotor-type agitator revolving inside of a cylindrical tank or cell, with baffles mounted around the periphyry of the latter. Using an apparatus of this type, it is possible to scrub the mineral particles by repeated collision of the particles. In cases where the mineral particles may be relatively soft, and the adhering slime or stains are difficult to remove, the addition of a hard, heavy grinding medium such as garnet or zircon may be used. Although Norman and Ralston do not mention it, the use of magnetite or ferrosilicon suggests itself for this purpose, since these scrubbing aids could be readily recovered by magnetic separation from the pulp after completion of the scrubbing operation, and reused.

Other types of apparatus such as blade mills, log washers, washing trommels, vibratory screen scrubbers, and so on, are also employed, or have been proposed, to scrub mineral surface.

A special type scrubber and mud ball disintegrator has been developed by Mead and Maust, formerly associated with Cyanamid's Florida phosphate operations at Brewster. This disintegration apparatus is described in U.S. Patent 2,297,009 and is being used with outstanding success in connection with the flotation operations at Brewster.

In many instances, a combination of chemical treatment and attrition scrubbing is necessary to prepare the mineral surfaces for subsequent contact with reagents required for flotation. Numerous examples are to be found where such treatment has yielded beneficial results, both in the laboratory and in large scale operations. For instance spodumene ores respond well to this treatment, as has been well set forth in a paper by Norman and Gieseke.[11]

These investigators found in their studies of spodumene rocks from six entirely different localities in the United States that:

> . . . difference in behavior of these rocks disappeared after proper cleansing of mineral surfaces and that after this cleaning, spodumene could be froth floated from associated minerals by the use of a number of different reagents.

The cleaning treatment, which varied in intensity according to the nature of the particular rock (the more weathered the material the more strenuous the treatment), consisted of vigorous agitation or attrition of the ground rock in a slurry containing a dispersant, such as sodium hydroxide. After proper cleaning of the crude rock the same flotation treatment, even to approximately the same amount of collector, gave highly selective flotation of spodumene. Oleic acid was considered the best collector for spodumene, from the standpoint of cost and efficiency.

In a recent U.S. Bureau of Mines report[12] by F. D. Lamb, results of investigations on New England beryl ores have shown that these ores were amenable to flotation only after desliming and vigorous surface conditioning with a suitable reagent such as caustic soda, trisodium phosphate, or hydrofluoric acid. Following such pretreatment, fatty acids in the form of emulsions were found to be most satisfactory promoters for beryl.

The benefits obtainable through the use of scrubbing and desliming procedures are well exemplified in the treatment of "Nelsonite" ore at Cyanamid's concentrator and pigment plant at Piney River, Va. At this property, "Nelsonite" ore (a geological term used to describe the weathered ilmenite ores of the district) is treated by flotation for recovery of ilmenite, from which is produced high quality pigment in an adjacent plant. The ore presently being mined is composed of ilmenite associated with apatite, ferromagnesian silicates, and quartz; and contains considerable amounts of kaolin, talc, amphibole, and iron oxide slimes. In order to produce a high grade ilmenite concentrate suitable for pigment manufacture, special procedures are required in the flotation plant to provide essentially complete removal of gangue slimes, and subsequent elimination of apatite in a two-stage flotation process. In this connection, it has been found that as little as 0.5 pct weight of gangue slimes affects flotation to the extent that twice the amount of promoter reagents normally used in treatment of clean ore is required for maximum recovery of ilmenite. In addition to the greater consumption of reagents, the presence of slimes interferes with selectivity.

In order to cope with this slime problem at the Piney River plant, it has been found necessary to adopt a flowsheet comprising initial desliming of the ore after crushing, and further desliming following each of three stages of grinding. This procedure gives effective elimination of slimes and minimum loss of ilmenite with the slimes. Under this system, the grinding units act as scrubbers in addition to freeing ilmenite from gangue.

Other examples include scrubbing and desliming of Florida pebble phosphate, beach sands, kyanite ores, glass sands, and others.

Although these procedures of attrition scrubbing and desliming are not as commonly employed in the treatment of sulphide ores as they are for nonmetallic ores, it is suggested that they warrant more attention than they have received. For instance, in the treatment of sulphide ores containing talc, graphite and similar soft minerals that tend to smear sulphide surfaces during the fine grinding stage, there might be benefits from an attrition scrub in the presence of dispersing agents ahead of further conditioning and flotation.

Discussion of the Principal Variables Involved in "Conditioning" of Previously Prepared Mineral Surfaces

The following interrelated variables (not necessarily in the order of their importance) influence the end results which are obtained when mineral surfaces are pretreated before flotation:

1. Time of treatment.
2. Number of stages of treatment.
3. Percentage of solids in the pulp.
4. Intensity of treatment.
5. Degree of aeration.
6. Temperature.
7. Method and order of addition of reagents employed for flotation.

TIME OF TREATMENT

Time of treatment is important. Too short an interval will not yield desired results. On the other hand, excessive time will result in increased consumption of reagents and power. Frequently, it also may be decidedly detrimental from the standpoint of formation of slimes which interfere with subsequent flotation, or because of destruction of beneficial films on the mineral surfaces.

In the treatment of nonsulphide base metal ores, little or no "conditioning" is usually desirable because the film formed by sulphidizing agents is readily destroyed or transformed.

In the final analysis, the time factor is related to pulp density, temperature of pulp, solubility of reagents, intensity of agitation, speed of reaction of reagents with mineral surfaces, and the state of the mineral surface.

As regards the flotation of quartz, and certain other silicate minerals, with positive ion promoters such as the long-chain amines or quaternary ammonium compounds, Dean and Ambrose[13] state that: "the highest recovery always is obtained . . . by stage addition; that is, an excess of reagent is never permitted in solution." Their experiments show that there is a rapid loss in efficiency when the time of contact between mineral and promoter extends beyond one minute. They suggest that the emulsifying power of the reagents removes water-repellant coatings from the particles if the solution is allowed to become sufficiently concentrated. These observations and experimental results have been confirmed in Cyanamid's Mineral Dressing Laboratories at Stamford, Conn., and at Brewster, Fla. That is little, or no, conditioning, and minimum agitation of feed after contact with the cation promoter, yields best results. In this connection, these facts led to the development of a special technique designated "flash flotation"; and a special flotation machine to put into practice this idea was designed by Mead and Maust, (formerly connected with Cyanamid's Florida phosphate operation) which is now covered in U.S. Patent 2,304,270. Broadly, their invention includes a process which involves the introduction of the ore pulp (previously contacted with reagents) into a flotation cell below the froth area but above the zone of agitation whereupon the ore is subjected to aeration with a minimum of agitation. The floatable portion of the ore material is immediately carried into the froth by the air bubbles, and the tails or nonfloatable material sinks until met by an upward flow of water from the

zone of agitation and is discharged from the cell at a point immediately above the zone of agitation. During the flotation operation the major portion of the ore pulp does not come in contact with any moving mechanical parts and hence there are produced no interfering secondary slimes which are so detrimental to flotation with positive ion reagents.

In the treatment of zinc sulphide ores, it is of interest to note, in connection with this discussion of the time factor, that, according to Mortensen,[14] the time required to activate zinc sulphides with $CuSO_4$ increases with the iron content of the blende and increases with rise in temperature and pH.

NUMBER OF STAGES OF TREATMENT

Number of stages of "conditioning" depends on the type of flotation separation desired. Normally, at least two tanks are desirable, to avoid short-circuiting. Two or more tanks or cells also permit flexibility in order of addition of reagents, adjustment of pulp density if desired, and variation in intensity of action if required between one stage and the next, and so on. However, if the "conditioning" operation is a simple one, and conservation of space is essential, a single tank may suffice.

PERCENTAGE OF SOLIDS

Pulp density is a critical factor in connection with the treatment of many types of ores—particularly in nonmetallic flotation. Coarse mineral pulps are more easily maintained in suspension at high solids than at low solids. In "conditioning" sulphide ore pulps it is usually desirable to conduct this operation at high solids because of greater concentration of reagents per unit volume of pulp and conservation in size of equipment required. For example in the activation of zinc sulphide, 1 lb of copper-sulphate per ton of ore at 50 pct solids means 1 lb of copper-sulphate per ton of solution whereas at 20 pct solids the quantity of copper-sulphate per ton of solution is only 0.25 lb. In addition, the volume occupied by 1 ton of pulp at 50 pct solids is less than ⅓ of the volume occupied by 1 ton of pulp at 20 pct solids. In other words, conditioning apparatus would have to be more than three times as large to provide the same time of contact of reagents with the pulp at 20 pct solids as it would for pulp at 50 pct solids.

Some slimy types of ores, particularly those containing bentonite, cannot be conditioned at high solids because the pulp becomes too viscous for proper dispersion of reagents, especially if the latter are poorly soluble or viscous themselves.

In the flotation of many nonmetallic ores with anionic-type promoters such as fatty acids, or soaps, greatly improved results are obtained, also savings in reagents can be effected, if the reagents are conditioned with pulp at high percentage of solids. This is particularly true when treating coarsely ground products. Outstanding examples of the minerals whose recovery is benefitted by such treatment, include: Florida pebble phosphate (collophanite), garnet, iron oxides, ilmenite, coal, feldspar, spodumene, brookite, rutile, kyanite, corundum, beryl, potash minerals.

Table 1 illustrates the beneficial effect of high solids conditioning in the treatment of a garnet ore.

Table 1 . . . Effect of High Solids Conditioning in Treatment of Garnet Ore

Test Number	Feed, Pct Garnet	Clean Conc., Pct Garnet	Rougher Tail, Pct Garnet	Pct Garnet Recovery in Cleaner Conc.	Pct Solids in Conditioning	Total Time of Conditioning
A	10.0	98.0	0.2	94.2	80	25
B	8.7	97.5	3.5	57.6	22	25
C[a]	8.9	97.6	3.5	43.8	22	25
D	10.7	98.4	0.8	88.5	80	10
E	10.0	54.0	0.2	85.5	22	None
F[b]	10.8	98.4	0.6	90.8	80	25

[a] Pulp aerated during conditioning.
[b] Acid and promoter added together.

Flotation feed was ground to minus 48 mesh and deslimed prior to conditioning.

The benefits of high solids conditioning in the treatment of ilmenite in a slime table tailing were reported in a recent interesting paper[15] by F. R. Milliken, presented at the February 1948 meeting of the AIME in New York. The author states that laboratory testing indicated that improved results were obtainable by greater intensity of agitation, and that in order to obtain this a shearing-type impeller with maximum circulation through the impeller was required for good ilmenite recovery and gangue rejection. This was confirmed to a considerable extent in the plant, after installation of a receded disc impeller and diffuser plate, also a central standpipe to provide maximum circulation in the conditioners. Although laboratory testing had shown no metallurgical benefits from conditioning at higher solids than 40 pct, it was found in the plant that when higher density was tried a further improvement occurred. With the high density conditioning, at about 63 pct solids, it was found that a simple impeller with four radial blades set at 45°, operated to give a downward thrust, gave satisfactory results. These impellers, 24 in. in diameter, operate at 260 rpm in 5 ft diam by 4 ft tanks.

Improved methods for removal of impurities from glass sands, ores, and others, are claimed in T. Earle's U.S. Patents 2,106,887 and 2,106,888 (Feb. 1, 1938). One of the features of these patents involves the preliminary conditioning with reagents and a controlled amount of water "sufficient to satisfy the affinitive capacities of all of the separated particles."

High solids "conditioning" does not appear to be necessary, or even desirable, in the treatment of such nonmetallic minerals as fluorite, calcite (in finely ground cement rocks) rhodocrosite, celestite, gypsum, graphite, talc, when these minerals are finely ground.

Likewise, in the flotation of quartz and certain other silicate minerals using cationic type promoters, it may or may not be advantageous to contact the promoter with the pulp at high solids.

INTENSITY OF TREATMENT

Some pulps require only sufficient time for completion of reaction. Others need, in addition to time, the catalyzing effect of intensity, as noted above.

Some pulps need the minimum of intensity, for example, soft ores which tend to form slimes, also ores which require sulphidization.

The type and intensity of conditioning can have a marked influence on the separating efficiency obtained in the subsequent flotation operations. Hence, it is vitally important in laboratory testing to try to simulate the action

of equipment that is either in use in the plant or which is commercially available.

An interesting example of the importance of intensified conditioning is furnished in an excellent article[16] by Messner and Bein which describes the Big Bend, Calif., operations of Hoefling Brothers. The original conditioners installed in this plant were equipped with 1½ hp motors which proved to be not large enough to permit good aeration. When plant results did not meet those indicated by laboratory testing it was realized that the aeration obtained in the Fagergren laboratory cell was much greater than that in the underpowered conditioners. New 5 hp motors were put on the tanks, the blades on the agitator were steepened to a 40° pitch and the agitators were speeded up to 300 rpm. The tailing losses immediately dropped after these changes. The authors state that increased pulp densities also may have contributed to reduced tailing losses.

At another California operation treating a high pyrite, zinc-lead ore containing 2.7 pct Pb, 9.5 pct Zn (about 3 pct nonsulphide) it was found that increased agitation and aeration during conditioning of the lead tailing resulted in marked improvement in grade of zinc concentrate produced in ensuing flotation operations, as shown by Table 2.

Table 2 . . . Effect of Increased Agitation and Aeration on Ensuing Flotation Operations

Product	Pct Pb	Pct Zn	Pct Zn Recovery	Type of Conditioning
Zinc concentrate	0.5	48–50	±67	Ordinary agitation and poor aeration.
Zinc concentrate	0.5	+60	±67	Vigorous agitation and strong aeration.

The major diluent in both concentrates was pyrite. Less than 2 pct insoluble was present. Increased agitation and aeration improved the rejection of pyrite.

DEGREE OF AERATION

Some pulps, particularly those of heavy sulphide ores, require intense aeration during the pretreatment stage. Certain other pulps do not require aeration and such action may be actually detrimental from the standpoint of increased reagent consumption, lowered flotability, and so on.

In plants where the flotation machines are "sluggish" and give poor aeration, intense aeration of the pulp prior to flotation frequently produces improved results.

The effect of aeration during conditioning has been the subject of a number of technical articles.

O. C. Ralston and his coworkers seem to be among the first to recognize the benefits resulting from adequate aeration during grinding and conditioning prior to flotation of certain types of sulphide ores. Their interesting investigations are reported in a technical paper.[17] Among other things, they concluded that:

> . . . grinding of an ore in limewater with exclusion of oxygen, produced a pulp which gave worse results in flotation of chalcopyrite and sphalerite and rejection of pyrite than one in which plenty of oxygen was present during grinding. [Also] . . . the addition of lime to a thick pulp from a grinding mill and conditioning by aerating permitted better subsequent flotation results than adding the same amount of lime to a pulp diluted to flotation density before similar conditioning.

In a written discussion[18] of this paper, C. G. McLachlan, Superintendent of Concentration, Noranda Mines Ltd., reported beneficial results obtained at Noranda, Quebec, by aerating ground flotation pulp prior to flotation in the presence of soda ash rather than lime. This is discussed more extensively in later articles by McLachlan.[19,20]

McLachlan considers that whenever aeration of pulp before flotation is necessary that:

1. Aeration and flotation steps should be carried out separately.
2. Depth of pulp in the conditioner is important if aeration is to be completely effective (more air is entrained in a deep pulp than in a shallow pulp).
3. The aeration step should be completed prior to flotation. One should not have to rely on a flotation machine to provide the remaining aeration that may be necessary if the pulp has not been aerated properly in the conditioner.

At Tennessee Copper Company's operations at Copper Hill, Tenn., J. F. Myers reports[a] that:

> . . . where both pyrrhotite and pyrite are to be depressed with lime, deep aeration conditioning and flotation must be done together. That is the reason we use the deep Munro-Pearse cells on our copper circuit.

[a] Personal communication.

Commenting on the action of the Munro-Pearse deep cell, S. R. Zimmerley of the Salt Lake Station, U.S. Bureau of Mines states[a] that:

> . . . an analysis of where pretreatment or conditioning leaves off and flotation begins might be pertinent. For example, in the deep cell air is introduced many feet below the surface of the pulp, whereas in Booth's new machine flotation is accomplished on introducing the air only a few inches below the surface. Apparently the deep cell also functions as a sort of dynamic conditioner. The individual mineral surfaces may be undergoing a change (involving some complex oxidation reaction perhaps) from unfavorable to optimum back to unfavorable conditions for flotation and when the most favorable condition exists for the individual mineral particles it floats and is removed from the system. If differently sized particles react at different rates, then any static or bulk system of conditioning would result in over-conditioning of some grains and under-conditioning of others. Deep cell flotation then would represent another fundamental type of conditioning.

Aeration of pulp in a conditioner prior to flotation was practiced in the treatment of the chalcopyrite-pyrrhotite ore milled by Consolidated Copper and Sulphur Co. at Eustis, Quebec. Milling practice at this old property which has subsequently been shut down, is reported in a paper[21] by Baxter and Snow.

As an interesting sidelight on this subject of aeration of pulps during conditioning, it is to be noted that, at one plant at least, improved results are reported to have been obtained by aerating the milling water. An article[22] by Owen Mathews, describes the benefits of this method of treatment in the flotation plant of Cochenour Willans Gold Mines Ltd. in the Red Lake area of Northwestern Ontario.

TEMPERATURE

Flotation results are usually improved when operations are conducted at elevated temperatures. This is particularly true when treating minerals of poor flotability such as marmatite. Also, the flotability of most nonmetallic minerals is considerably poorer in cold water than in warm water when soap-type promoters are used. Under certain

[a] Personal communication.

favorable conditions, particularly where it is possible to pretreat pulps at high solids it may be a paying proposition to heat the water used for the pretreatment step. The costs for such treatment may be offset by reduction in reagent requirements, reduced amount of equipment for pretreatment and flotation, or better overall metallurgical results. Heating of reagents before adding to mill pulps is also reported by J. J. Burns, to be beneficial at the Balmat, New York, mill of St. Joseph Lead Co.

In the early days of flotation, heat was commonly used in the conditioning stage ahead of flotation of zinc sulphides. Currently, few plants employ this aid. Those that do maintain that the benefits of heat cannot be realized by substituting reagents, time, or intensity of "conditioning." Heat seems to be particularly valuable, especially in wintertime operation when treating martite ores. Copper ammonium sulphate is reported by D. M. Kentro to give better results than copper sulphate and has replaced heating of pulps at the Shenandoah Dives mill in Colorado.

Some nonmetallic operations report benefits from heating of pulps prior to flotation. The selective flotation of fluorspar from other easy floaters such as calcite is aided partly because less promoter (oleic acid) is required in warm pulps compared with cold pulps.

Wark and Cox have made laboratory studies of the influence of temperature on adsorption of xanthate at surfaces of certain pure sulphide minerals—galena, pyrite, arsenopyrite, chalcopyrite and sphalerite, and on the effects of alkali, cyanide and copper sulphate in hindering or promoting the adsorption. Their conclusions as expressed in their paper[23] are as follows:

1. In the absence of copper sulphate, a change in temperature from 10 to 35°C merely alters slightly the amounts of depressants (cyanide and/or alkali) necessary to prevent contact, a greater concentration of depressant being necessary to prevent contact at 10° than at 35°C.

2. In the presence of copper sulphate, the influence of temperature on the amounts of cyanide and/or alkali necessary to prevent contact is slight for chalcopyrite and sphalerite, but is relatively great for pyrite and arsenopyrite. Under these conditions, pyrite and arsenopyrite are much less influenced by the depressants at 35° than at 10°C.

3. The conditions most favorable for floating sphalerite away from pyrite are low temperature, low xanthate concentration, and if low temperature is impractical, a small range of pH values which for the conditions used here is between 6 and 8.

4. The conditions most favorable for floating chalcopyrite or other copper-bearing sulphide minerals away from pyrite are likewise low temperature and low xanthate concentration.

5. There appears to be little advantage in raising the temperature of the system above 35°C when pyrite or arsenopyrite is to be floated in the presence of copper sulphate and sodium cyanide.

6. The differentiation between sphalerite and pyrite when using amyl xanthate in a circuit containing copper sulphate and cyanide is much diminished as the temperature is raised.

It is to be noted that certain of these conclusions would appear to be at variance with results obtained in practical operating practice.

While on the subject of temperature, mention should be made of the utilization of steaming of pulps of copper sulphide-molybdenite concentrates, to destroy the promoters and permit superficial oxidation of the copper sulphides preparatory to selective flotation of molybdenite.

The use of high temperature conditioning (above 35°C and preferably above 60°C) in connection with the soap flotation of nonsulphide ores, as covered in B. Kalinowski's French Patent No. 847,215, Dec. 7, 1938, is claimed to provide more intensive activation, better separation of values from gangue, and reduction in quantity of reagents.

It is reported that at one of the fluorspar flotation operations in this country enhanced results are obtained when the flotation feed is conditioned at boiling temperature for a period of 15 to 20 min prior to flotation without cooling the pulp.

METHOD AND ORDER OF ADDITION OF REAGENTS

In general the desirable sequence of addition of reagents seems to be:

1. Regulating agents—lime, soda ash, acid, and others.
2. Activating or depressing agents.
3. Promoters and frothers.

Although the above generalization will apply to many ores, there is no universal acceptance among millmen as to the proper sequence of reagent additions. One can usually start an argument at any meeting of practical operators on such controversial points as the following—to name a few:

1. Addition of lime before or after copper sulphate for zinc sulphide flotation. Also the employment of alkalinity regulators before or after addition of activators for other minerals.

2. Addition of depressants before or after addition of alkalinity or acidity regulators.

3. The benefits obtainable when promoters are added together with, prior to, or following, the addition of depressants, activators, or regulating agents.

4. Whether some reagents can be most effectively used when dispersed, solubilized, diluted, or mixed with the mill water before being added to the pretreatment apparatus. For example, the addition of emulsified fatty acids to mill water, cation reagent to mill water, or dilution of concentrated sulphuric acid before adding to pulp.

In general, it is usually desirable to try to complete one reaction with one reagent before another reagent is added. Reagents should not have to compete with one another, at the expense of reacting with mineral surfaces.

F. W. McQuiston, Jr., notes[a] that the unorthodox practice of adding copper sulphate and xanthate in stream contact with each other for zinc sulphide activation and promotion has been found to be the most effective method of reagent addition used to date at an important western mining operation. He adds that the copper-xanthate precipitate formed is a potent sphalerite promoter.

An example of the importance of proper sequence of reagent addition is to be found in the treatment of complex copper-zinc ores of California where the order of reagent addition in conditioning ahead of flotation has a marked influence on selectivity and recovery of values.

The following practice is reported to yield best results on ores containing the copper minerals chalcopyrite, chalcocite, covelite and bornite; and sphalerite and pyrite; with gangue minerals, comprising quartz, chlorite, sericite, amphibole, and barite.

Copper Recovery

1. In the grinding circuit, excess alkalinity is avoided, and the pH kept at about neutral or below 7.3. Sodium

[a] Personal communication.

sulphite is added to the ball mill as an auxiliary depressant for sphalerite and pyrite.

2. To the conditioner ahead of flotation, cyanide is added as a zinc depressant, together with a small amount of an auxiliary promoter for copper sulphides.

3. To the flotation circuit a small amount of the principal promoter and a frother are added to assist recovery of the copper sulphides.

4. The rougher concentrate is cleaned after conditioning it with cyanide and zinc sulphate. The cleaner tailing is returned to the grinding circuit.

Zinc Recovery

The tailing from the copper roughing circuit is treated as follows:

1. To the primary conditioner copper sulphate and a suitable promoter are added. It is important to avoid alkalinity at this stage and the pH of the pulp is about 6 to 7.

2. To the secondary conditioner lime is added to give an alkalinity corresponding to 0.5 to 1.0 lb CaO per ton of solution.

3. Following this two-stage conditioning an auxiliary promoter and frother are added and a zinc concentrate is recovered.

4. The zinc rougher concentrate is cleaned and the cleaner tailing returned to the primary conditioner.

General Notes in Connection with the Above

Proper alkalinity at the right time and at the right place, plus the right order of addition of copper sulphate and lime to the conditioners are all key factors in securing optimum results. Good aeration during conditioning is also helpful in securing selective flotation. The double conditioning in the zinc circuit is especially valuable in wintertime operation when low pulp temperatures obtain.

In the flotation of some oxidized lead ores, a combination of sodium sulphide and copper sulphate yields better results than sodium sulphide alone. When these reagents are used, the order of addition is important. The sulphidizing agent is added to the pulp first and conditioned with it for only a short period—3 to 4 min—and then copper sulphate is added in a separate conditioner to complete the activation and neutralize excess causticity due to the sodium sulphide. Finally, the promoter agent and frother are added to the conditioned feed just ahead of the flotation machines.

A. F. Taggart notes[24] that "Unlike sulfides, the nonmetallic minerals will not, in general, float well without the presence of an insoluble coating oil in the pulp." For this purpose he states that petroleums are generally used and that the sequence of steps in oiling are first, reaction between the organic acid radical of the collector and the mineral, to form a water-insoluble coating compound and thereafter oil-coating of the altered surface. He states that:

> Addition of the collector agent in solution in the insoluble oil has the further advantage that it holds back saponification of the fatty or resin acid and thus introduces the organic acid radical in ionized form and at a rate, substantially, no faster than the mineral can consume it. There is thus, little wastage of the fatty acid, and the evil effects of excess soluble soap (overfrothing) are avoided.

The *place* at which reagents are added, as well as the order of their addition can have an important bearing on the efficiency of separation in the subsequent flotation step. In this connection, certain experiments by Gaudin and Malozemoff on sulphide minerals of near-colloidal size, results of which are reported in a technical paper,[25] lead these investigators to postulate two principal reasons for the poor flotation of minerals of near colloidal size. In brief, sulphide minerals with "young" surfaces do not float readily, for mechanical reasons, that is, low probability of contact of the particles with bubbles, and sulphide minerals with "old" surfaces do not float readily because they have had their surfaces exposed for a long time to the action of salts in the pulp so that their surfaces become coated with an inert compound. Since, owing to their size, they do not become ground while the coarse particles of ore go through the grinding steps, their behavior in flotation is subnormal. However, experiments by these investigators indicate that the formation of these coatings can be completely prevented by the addition of the collector into the grinding machine so that the collector shall be present in the grinding system while the mineral particles are being broken and expose their fresh surfaces. In this way, the authors claim, the collector is permitted to cover the surface of the mineral first and thus protect it against other agencies in the system. These latter conclusions seem to be at variance with the postulation that because of their fine size the particles with old surfaces do not become ground, and it would appear that there is no reason why they should be affected by the promoter added to the grinding circuit. It is quite conceivable however, that in the case of freshly-ground sulphides of near colloidal size it would be advantageous to have promoter available for reaction as soon as the fresh surface is exposed, thus preventing reaction between the fresh surface and soluble salts in the pulp. However, it would also appear that the addition of promoter to the grinding circuit would not be justified if two sulphides of near flotability were required to be separated in a subsequent flotation step. In the latter case, the presence of the promoter might nullify the effect of the depressant, or depressants, which was added to insure selective flotation.

Apparatus Used for Pretreatment of Mineral Surfaces

It is beyond the scope of this paper to describe in detail the various types of equipment that are in use, or which might be employed, for the pretreatment of mineral surfaces ahead of flotation. However some brief remarks on this subject may be of interest.

As regards scrubbers, and attrition mills, passing reference has already been made in this paper to certain types which provide for more intense cleaning of mineral surfaces than the better-known forms of disintegration apparatus such as the drum scrubber, blade mill, log washer, sand screw washer, trommel washer, and the like.

The purposes of "conditioning" apparatus have already been stated. The type and size of these units will depend upon the functions which they are called upon to perform.

For simple conditioning of finely ground pulps at relatively low solids (20 to 50 pct) a round tank provided with means for positive circulation of pulp at sufficient velocity to prevent sedimentation of solids, may be used.

When finely ground pulps are to be handled at high solids or where more intensive agitation and aeration of pulps is required, round tank-type conditioners may be used—but special type agitators, such as turbine-bladed impellers, operating at relatively high speed are desirable.

In cases where intense aeration is

essential, the special type of aerator developed at Noranda Mines Ltd. may be required. A description of this apparatus is given by Taggart.[26] It comprises a circular tank about 9 by 15 ft with four rubber air-lift pipes, a plurality of radial air-inlet pipes of different inward extension, and a slow-moving under-driven rake mechanism at the bottom.

For the treatment of coarse, relatively slime-free pulps, and fast-settling pulps of heavy minerals, at high percentage of solids, it is common practice to use a multiple of shallow cells. For example, for the preoiling of relatively coarse, deslimed phosphate flotation feed at 69 to 70 pct solids, the apparatus used at one plant comprises a four-cell unit, each cell 43 by 43 in., with provision for an effective pulp depth of 22 in. Cruciform impellers used for mixing are 18 in. diam with 4 in. face, and turn at 160 rpm. Actual power consumption in the operation of these impellers amounts to 10 hp per cell. At another nonmetallic operation, three 31 by 31 in. cells, providing an effective pulp depth of 24 in., are used to preoil at 67 pct solids a coarsely ground deslimed ore of about 2.8 sp gr. The rubber-covered propeller-type agitator, 12 in. diam, is run at 235 rpm, and in a direction to provide a down thrust to the pulp. Power consumption amounts to 7½ hp per cell.

At still another nonmetallic operation, a minus 35 mesh, deslimed ore of relatively high specific gravity—3.2—is "conditioned" at approximately 80 pct solids in two 4 ft diam by 3½ ft depth wooden tanks equipped with 24 in. diam turbine-blade type agitators. The latter are driven at 163 rpm. Power consumption amounts to 7½ hp per unit.

Drum-type mixers, revolving on trunnions, and equipped with lifter blades are also very effective for mixing coarse, deslimed phosphate flotation feed at high solids with reagents.

Summary

This important phase of the flotation process—pretreatment of mineral surfaces—warrants more attention than it sometimes receives; since it offers opportunities for improved metallurgical results as well as savings in reagents and flotation machines. Laboratory control tests on mill pulps should conform to the flowsheet and type of apparatus used in the plant for pretreatment or "conditioning." In laboratory studies close attention should be paid to the variables involved in pretreatment of mineral surfaces. Flowsheets and apparatus recommendations for commercial operation based on such studies should conform to the particular procedures and techniques developed in the laboratory, in order to ensure maximum dollar return at minimum cost.

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Open Fracture in Langbeinite, International Minerals and Chemical Corporation's Potash Mine, Eddy County, New Mexico

By JAMES B. CATHCART,* Junior Member AIME

The potash mine of the International Minerals and Chemical Corp. is about 18 miles east of Carlsbad, New Mexico, in sec 1 and 12, T 22 S, R 29 E, N.M.P.M. Potash is produced from two zones in the Salado formation of late Permian age. The lower zone, consisting of a mixture of sylvite (KCl) and halite (NaCl), lies at a depth of about 900 ft, and the upper zone, in which the principal potassium mineral is langbeinite ($K_2SO_4 \cdot 2MgSO_4$), lies at a depth of about 800 ft. Because of the plastic nature of the material, fractures in the Salado formation are extremely rare and have been noted only on the 800 ft level of the International mine. The open fracture described below is almost unique. Only one other open fracture has been noted, and it was on a working face in the same general area as the one described, but was destroyed before the writer had a chance to examine it. According to J. E. Tong, foreman on the 800 level, this fracture was similar to, but smaller than, the one herein described.

The saline strata are nearly flat-lying and comprise interbedded layers of halite, anhydrite ($CaSO_4$), and polyhalite ($2CaSO_4 \cdot MgSO_4 \cdot K_2SO_4 \cdot 2H_2O$), with halite predominating. The halite layers contain local concentrations of sylvite, carnallite ($KCl \cdot MgCl_2 \cdot 6H_2O$), and langbeinite. The principal marker in the potash mines area is a bed of anhydrite about 8 ft thick, and 70 ft below the 800 level. This bed contains varying amounts of polyhalite, and locally a few blebs of sylvite.

Two principal marker beds are present over most of the 800 level. The lower is the "Middle Salt Marker," a bed of bright-orange halite, which ranges in thickness from a knife-edge to about 0.6 ft, and is usually bounded above and below by thin seams of green or reddish-brown clay. In langbeinite-rich areas, this marker usually contains abundant blebs of langbeinite. The upper of the two principal markers is a green clay seam, containing abundant halite, which ranges in thickness from 0.1 to about 0.5 ft. It is 2 to slightly more than 5 ft above the Middle Salt Marker. No langbeinite of minable grade is found above this clay, and it

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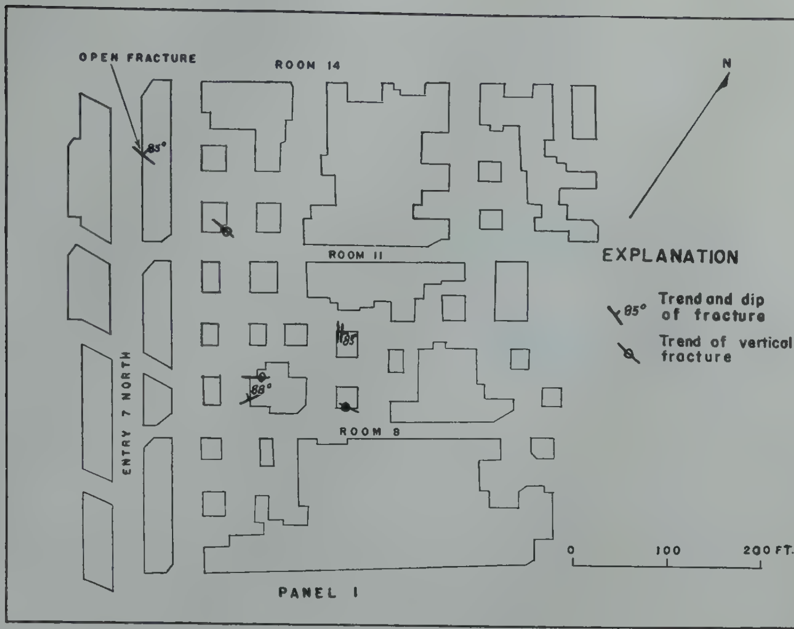

FIG 1—Plan map of a portion of the 800 level. (Base from Company map, April 1947.)

forms a comparatively even roof, so in the ordinary course of mining the salts are broken to this clay seam. A third, but less prominent marker, the "Lower Salt Marker," is also present on this level. It is a bed of orange halite, but is indistinct, and it is not present, or at least not recognizable, over much of the level. Where it can be recognized, it is about 2 ft below the Middle Salt Marker.

The open fracture in the langbeinite is found on the east wall of entry 7 north, between rooms 11 and 14 (Fig 1). The fracture strikes N 85°E and dips 85° north. The opening along the fracture extends 2 ft vertically, is 0.7 ft in maximum width, and extends at least 2.5 ft into the barrier pillar. About 1 ft back of the face of the pillar, the width of the opening is less than 0.1 ft. It is probable that the actual maximum width is somewhat less than the 0.7 ft, because the opening has been enlarged by blasting and subsequent chipping. Nothing is known of the former extent of the opening into the area now occupied by the entry. The fracture does not extend across the entry, however, as no trace of it can be found on the pillar on the opposite side of the entry. The open part of the fracture is confined to the highest grade langbeinite, and pinches down to a knife edge above and below. Above the langbeinite is a mixture of sylvite and halite, and the fracture stops abruptly at the contact. Below the opening, the fracture can be traced down through the Middle Salt Marker, which is here chiefly langbeinite, to the lower contact of langbeinite and halite, where it is sharply cut off.

The sharp termination of the fracture at the contact of the langbeinite with mixed sylvite and halite above, and with halite below is particularly striking (Fig 2). Langbeinite has a hardness (Mohs scale) of 3 to 4, whereas the hardness of halite is 2.5 and that of sylvite is only 2. Langbeinite is apparently the only mineral hard enough and competent enough to maintain an opening. Halite and sylvite may have been competent enough to fracture, but if so the weight of the overlying beds has caused these salts to flow slightly, healing the fracture. It is interesting to note that the fracture is sharpest and clearest in the best grade of langbeinite; that is, where there is least halite in the langbeinite. In the langbeinite below the Middle Salt Marker there is a large proportion of halite, and the fracture can be traced only with difficulty. The fracture does not extend to the opposite side of the barrier pillar, probably because the opposite side of the pillar consists almost entirely of halite, not competent enough to fracture.

Evidence that solutions passed through the fracture is lacking. The edges of the fracture are sharp and clear, and there is no evidence of solution or erosion of the wall rock. A very thin film, composed of a very fine-grained to microcrystalline white mineral, picromerite, a hydrate of langbeinite with the composition $K_2SO_4 \cdot MgSO_4 \cdot 6H_2O$, is present on the walls of the opening, and as a filling where the fracture narrows down. This mineral may have been precipitated from connate brine squeezed from the enclosing salts.

Evidence that this is a true fracture and not a crack caused by blasting can be summarized as follows:

1. The fracture is continuous through the langbeinite; blasting cracks usually extend only about a foot from the drill hole.

2. Openings along blasting cracks are narrow, none are more than a fraction of an inch wide, and none are very deep.

3. No coating of hydrate is found in the cracks produced by blasting.

4. Blasting cracks are normally confined to the corners or edges of pillars;

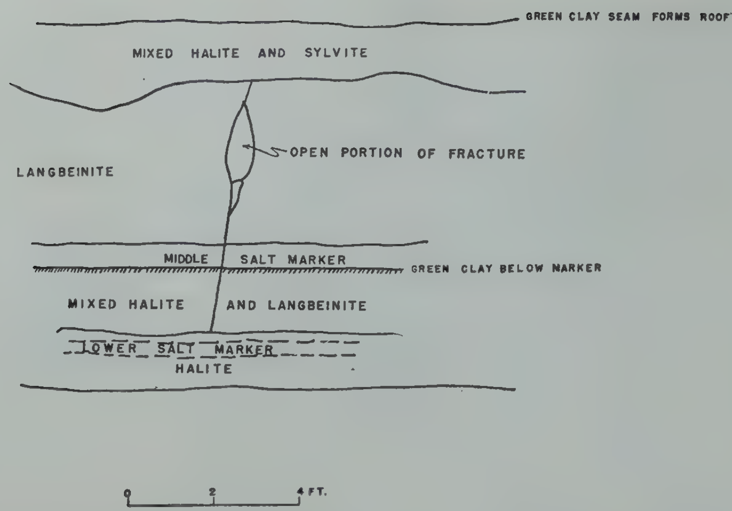

FIG 2—Diagram of open fracture in langbeinite, entry 7 north, 800 level.

this fracture is almost in the center of a pillar.

5. Blasting cracks are most numerous at the end of a round, that is, the bottom of the drill hole; this fracture is in the approximate center of a round.

Several other fractures are shown on the map (Fig 1). None of these fractures are open, but they are similar to the open fracture in that they occur in bodies of langbeinite, terminate at the top and bottom of the langbeinite, and have a thin film of picromerite on the fracture face. Thus, they are believed to be true fractures rather than blasting cracks. Probably more fractures are present than are shown, but those that are not open are difficult to distinguish from blasting cracks, and any that were of doubtful origin were omitted from the map.

Possible Origin

A slight regional tilt to the southeast, and superimposed broad, gentle, northwest-trending folds are the major structural features in the vicinity of the International mine. The fractures in the langbeinite bodies are possibly due to the same compressive stresses that caused the gentle folds.

Practical Aspects

It is of interest to note that heretofore it has been generally assumed that because of the plasticity of salt and sylvite there would be no fractures through which water and gas could enter the mine. The recognition that fractures do occur in the "salt" section is important in that it should influence the approach to such matters as the plugging of drill holes and other mining problems involving the exclusion of water from the salt.

It is of interest to note here that after this paper was written, a long entry of the Potash Company of America penetrated a crack through which oil was seeping. This exposure was not seen by the writer, but it serves to illustrate the importance of the fracturing in the "salt" section.

Salt Resources of West Virginia

By PAUL H. PRICE,* Member AIME, and JOHN P. NOLTING*

History

The history of the salt industry in West Virginia dates back nearly two hundred years; however, the history of salt as an important raw material for the chemical industry is much more recent.

The earliest record of West Virginia salt was in 1753, when a raiding party of Shawnee Indians attacked frontier settlements in Virginia taking captive a number of the pioneers. One of these, who later escaped and returned to her friends, described how the Indians camped at a salt spring on the banks of the Kanawha River and evaporated the water to obtain a supply of salt to take back to Ohio with them.

Settlement of the Kanawha Valley began in 1774, that same year the Battle of Point Pleasant ended the Indian's power in the Ohio Valley; and settlement of the Kanawha Valley progressed rapidly with a consequent increase in the importance of the Kanawha Licks as a source of salt. In those days the evaporation of the brine was a family matter for which the family wash-kettle was commandeered.

In 1797, Elisha Brooks erected the first salt furnace in the Kanawha Valley. It consisted of two dozen small kettles set in a double row, with a flue beneath, a chimney at one end, and a fire bed at the other. He obtained his brine at a depth of 20 to 30 ft by sinking hollow logs, called "gums" into the salt lick and dipping the brine with a bucket. His production was about 150 lb a day and sold for 8 to 10 cents a pound.

In 1806, the Ruffner brothers, David and Joseph, prospected for and drilled the first well in America using a spring pole and steel chisel bit. After several failures they succeeded, in 1808, in drilling to a total depth of 59 ft where they secured a good flow of strong brine. Most standard drilling tools, jars, casing, and practically all the basic oil well drilling machinery now in use, were developed in the drilling of salt wells. To evaporate this brine, the brothers built a wood-burning furnace similar to that of Brooks but with a better arrangement and with more and larger kettles. When production was started they reduced the price of salt to 4 cents a pound.

By 1817, there were some 30 salt furnaces in the Kanawha area. It was in this year that bituminous coal first was used as an industrial fuel. David Ruffner again pioneered the way by converting his furnace for the consumption of coal. This year also brought about the formation of the first trust which controlled production and set a price of 1 cent a pound for the salt.

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Table 1 . . . Summary of Averages of Chemical Analyses of Brines of West Virginia for Four Major Horizons, in Parts per Million[1]

Geological Horizon	Number of Samples in Average	Temperature °C	Density	Solids after Evaporation	Fe	Ca	Sr	Ba	Mg	Na	K	HCO_3	SO_4	Cl	Br	I	Total Determined Constituents
Salt	106	25.1	1.0221	60,089	215	4,284	128	316	888	16,840	167	279	23	36,401	210	2.33	59,380
Maxton	19	25.4	1.0597	87,642	42	6,274	168	679	1,459	24,063	204	132.3	7.9	52,934	318	3.7	86,288
Big Injun	33	25.6	1.0697	99,845	155	7,722	204	193	1,685	26,994	320	129	41	60,704	458	6.8	98,532
Oriskany	6	23.3	1.1936	252,717	60	17,615	1,010	32	2,606	72,317	2,777	127	342	152,870	1,135	11	250,730

[1] References are at the end of the paper.

In 1833, George H. Patrick invented the multiple effect evaporator. In this system the brine from the wells was first pumped to a reservoir, and the desired amount allowed to flow to the pan where it was concentrated almost to the saturation point. It then flowed to settling vats where the muddy material and iron salts settled out. After this the purified salt solution flowed to the "grainer" vat for further evaporation resulting in the finished product. Salt then sold for ½ cent a pound.

The use of gas as a fuel for the salt industry was brought about by accident. In 1841, William Tompkins, in boring a salt well near Washington Spring, struck a large flow of gas which he used as fuel for his furnace thus making a great saving in fuel and lowering the cost of production.

The period of maximum production of the Kanawha Valley salt industry occurred between 1845 and 1854, beginning with road improvements and ending with competition from Ohio Valley producers which further cut the price to ¼ cent a pound. The peak year was 1846, during which 3¼ million bushels, or 80,620 short tons, of salt were produced. By 1875, the ten existing salt furnaces had a capacity of about two and one-half million bushels but because of the competitive market less than one million bushels were produced.

During the time of the rise of the Kanawha salt industry minor salt developments took place in other West Virginia counties, but all of these were secondary in importance to the Kanawha industry.

The most important of the other salt developments was that in Mason County. This development was started at West Columbia in 1847, by three former Kanawha salt makers. The brines in this area were denser than the Kanawha brines and the area had the added advantage of better river communication with the West. By 1876, Mason County had a capacity of 75,000 tons of salt a year.

Salt was also produced in small quantities from wells scattered throughout the rest of the state. The production from these wells was so small that most of them deserve no more than passing mention. However, one well drilled near Sago, Upshur County, is of interest in that some years before the beginning of the oil industry, when the owner tried to deepen his well to increase production, he struck oil, and had to abandon the well because the unwanted oil ruined his salt production.

Despite the decrease in the importance of the Kanawha salt industry after 1854, some men recognized the possibility of using salt brine as a raw material for chemical products, as early as 1876. This did not materialize however, in West Virginia, until the early days of World War I.

In 1914, the Warner-Klipstein Chemical Co. erected a plant in South Charleston to produce chlorine and caustic soda from brine. This plant is now the Westvaco Chlorine Products Corp., the largest chlorine production plant in the world.

Between 1913 and the end of the war, many other chemical companies constructed plants to use salt brine as a raw material for the manufacture of chemical products.

Since that time the chemical industry, based largely on the use of salt brine, has experienced a tremendous growth in the Kanawha Valley.

Although, for many years, rock salt was believed to be present under the northern part of the state, it was not until the Second World War that the rock salt industry had its beginning in West Virginia. In 1942, the Defense Plant Corp. built an electrolytic caustic soda plant at Natrium, in Marshall County, where the rock salt is extracted by water solution from drilled wells. This plant is owned and operated now by the Columbia Chemicals Division of Pittsburgh Plate Glass Co.

Sources and Origin of Salt Brine

Salt occurs in two major forms in West Virginia, first as a salt brine and second as rock salt. Salt brine may be thought of as connate, or entrapped sea water.

As marine sedimentation took place solid material such as clay, silt, or sand particles, settled in standing bodies of water to form unconsolidated sediments, the pore spaces of which were filled with salt water. As the beds became thicker and compaction took place, some of this pore water was squeezed out and some remained in the more porous beds. As cementation occurred part of the original sea water was trapped in the pore spaces. In some cases, later land movements elevated these water-bearing rocks above sea level, erosion exposed the brine-bearing horizons permitting the entrance of surface waters which resulted in the dilution of the contained brines. In other cases, gas associated with brine-bearing rocks caused the evaporation of part of the water resulting in concentration of the brine.

Beside the dilution and concentration of brines, temperature and pressure changes, and contact with rocks, gases and other waters may all have had an effect upon their present composition.

In studies of the salt brines of West Virginia, carried on by the West Virginia Geological Survey in 1937 and 1947, brine samples were obtained from 21 different horizons in 38 counties of the state, and in adjacent parts of neighboring states. These horizons range in geologic age from the Moundsville Sand (Saltsburg) of the Conemaugh to the White Medina (Clinton of Ohio) of the Silurian. Complete chemical analyses of 308 samples have been made. A study of the producing horizons leads to the conclusion that the concentration of the brine increases with geologic age. The heaviest Salt

Sand brine taken from a well had a specific gravity of 1.1243; the heaviest Big Lime brine was 1.1299; the heaviest Big Injun brine was 1.1449; that for the Brown Shale, 1.1617; and the heaviest Oriskany was 1.2246. The brines from the lower horizons seem to be very nearly saturated with calcium chloride. Brines obtained from the anhydrite-gypsum zones of the Silurian are generally impregnated with hydrogen sulphide. The reserves of the four largest producing horizons, the Salt Sand, Maxton, Big Injun, and Oriskany, have been calculated as totaling nearly 800 million tons of salt.

The Salt Sands ordinarily contain the larger volumes of brine; the brine from the Big Injun is generally of greater density but the volumes are usually smaller.

Table 1 gives a summary of the average chemical analysis of the brines of the four major producing horizons.

Geologic Occurrence of Salt Brines

As brines are considered to be connate water, the search for them should be confined to marine sediments. The upper half of the Conemaugh and all of the Monongahela Series of the Pennsylvanian System and Dunkard Series of the Permian System are of continental or nonmarine origin. The youngest known marine formation in West Virginia is the Ames limestone which occurs in the middle of the Conemaugh Series, or about 300 ft below the well-known Pittsburgh Coal; hence the commercial occurrence of salt brines would not be expected above this horizon.

PENNSYLVANIAN SYSTEM

The Conemaugh Series

The lower half of the Conemaugh Series contains three brine-bearing horizons; however, they are relatively shallow and not of commercial importance.

The Allegheny Series

Although the Allegheny Series has several massive sandstones they are not important brine-bearing formations.

The Pottsville Series

The Pottsville Series contains several massive sandstones which produce brine throughout the oil and gas fields of the state. These horizons are referred to by the drillers as the "Salt Sands" and may be numbered according to their position in the rock column. Due to their widespread occurrence and porous texture they offer great commercial possibilities for brine production. It was this group of sands which made possible the development of the early salt industry in the Kanawha Valley.

MISSISSIPPIAN SYSTEM

The Mauch Chunk Series

This series, which is composed of red shales and thin sandstones in the northern part of the state, thickens to the south with a considerable increase in the amount of sandstone present. The lower part of the series, having more marine sediments, includes the Maxton sand which contains brine in northern West Virginia.

The Greenbrier Series

This series is predominantly marine limestone but produces some oil and gas, and brine has been reported from "Big Lime" wells in a few localities.

The Maccrady Series

This series consists essentially of red shales and is usually thin or absent in the oil and gas regions of the state. While it is not an important brine producer in West Virginia, it is the source of the salt produced at Saltville, Va.

The Pocono Series

The Pocono Series contains marine sandstones and shales. Among the several oil and gas sands which it contains, the "Big Injun" is the most important from the standpoint of both petroleum and brine. The "Big Injun" contains large quantities of very concentrated brine in some parts of the state.

DEVONIAN SYSTEM

The Oriskany Series is the only member of the Devonian which is an important brine horizon. It is definitely of marine origin and the brine from this series is much more concentrated than the brines from the younger formations.

SILURIAN SYSTEM

Although some few samples of brine have been obtained from Silurian rocks, these rocks are not considered as important brine bearers. The importance of this system from the viewpoint of salt lies in the rock salt deposits of the Salina Group.

Rock Salt

Records of wells drilled in western Pennsylvania and eastern Ohio indicated that the Silurian salt beds, long worked in New York, would be present under our northern panhandle. Columbian Carbon Company's Lewis Maxwell well in Doddridge County, was the first to show conclusively that such deposits actually existed. Since then several more wells have shown its presence; however, the great depth of the salt has retarded its commercial development. Where the salt is now recovered from mines and wells in New York State, it is less than 1500 ft below the surface; at Cleveland, Ohio, 2400 ft, and at Barberton, Ohio, 2800 ft. The shallowest depth in West Virginia is nearly 5000 ft.

The rock salt deposits of West Virginia and the associated dolomites, anhydrites, and limestones are stratigraphically equivalent to the Salina Group of New York and to the Wills Creek or Roundout limestone of the eastern counties of West Virginia. Part of the salt is probably of the same age as the Tonolaway or Bossardsville limestone of the eastern outcrops.

Except where the salt is actually mined as in New York and Michigan, our knowledge of the salt is based on drill cuttings and cores.

In drilling wells with rotary equipment, salt does not appear ordinarily in the cuttings from these great depths because it is dissolved in the drilling muds.

Anhydrite, the anhydrous calcium sulphate, is much less soluble in water than salt and less concentration of sea water is required for its precipitation. Hence, where bedded salt deposits occur, anhydrite is distributed more widely than salt. In the Appalachian region the driller is likely to record the occurrence of anhydrite as "lime," thus it may occur in many wells where its presence has not been recorded.

CHEMICAL COMPOSITION

The chemical composition of rock salt from three different localities in the state is essentially the same in so far as the soluble salts are concerned,

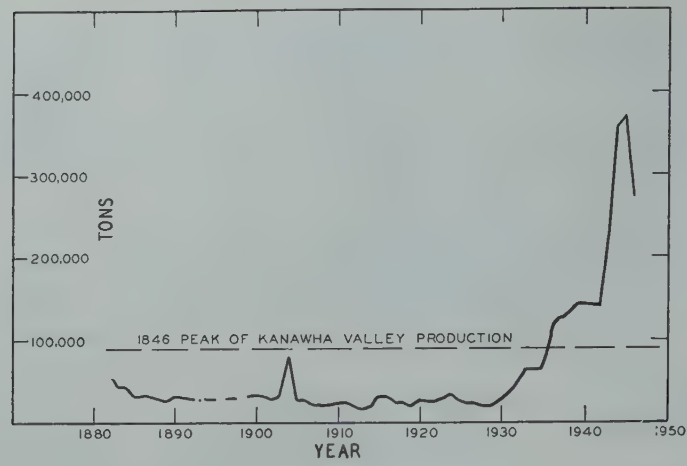

FIG 1—Salt production in West Virginia.

the chief difference being in the insoluble residues. While the brine from the solution of rock salt would need treatment before using, the amount of impurities is comparatively small. Sodium chloride is much higher in percentage in rock salt than in natural brines and calcium and magnesium salts are quite low. Bromine and iodine are trace elements, and barium is generally absent.

ORIGIN OF ROCK SALT

Nothing gained from our knowledge of rock salt deposits in West Virginia throws any further light on the existing theories of origin. There can be no doubt that the Silurian salt deposits of New York, Pennsylvania, Michigan, Ohio, and West Virginia were formed in the same general way, even though conditions varied in some detail in the various states.

It is generally agreed that bedded deposits of rock salt were formed by evaporation of salt water, where desert conditions prevailed; and there could have been no appreciable amount of fresh water flowing into the salt-water body.

Probably the most comprehensive work ever done on the subject was by H. L. Alling,[2] who, after due consideration of many theories of origin, came to the conclusion that: "The salt must have originated by evaporation under desert conditions behind a bar or barrier separating the basin or basins from the sea." The amount of salt in the sea water entrapped in the basin could not be sufficient to produce such thicknesses of rock salt as are known to occur; it is necessary then to consider a bar or barrier between the basin and the sea, low enough to allow the influx of additional sea water into the basin, but shallow enough to prohibit the outflow of the denser basin water to the sea.

The evaporation of the sea water resulted in the differential deposition of the various salts. As evaporation proceeded and the sea water was reduced to about 11 pct of its original volume only iron, calcium carbonate, and gypsum were deposited. As the volume decreased from 11 to about 1.5 pct, sodium chloride was precipitated along with small amounts of gypsum, magnesium sulphate and chloride, and sodium bromide. The "mother liquor," namely, the sea water less than 1.5 pct of the original volume, contained the remainder of the sodium chloride and most of the magnesium sulphate and chloride, sodium bromide and potassium chloride. This differential deposition coupled with the influx of additional sea water, resulted in the formation of large deposits of fairly pure sodium chloride. The reason for the nondeposition of the salts remaining in the "mother liquor" is probably that these dense solutions finally reached the level of the barrier and flowed out into the sea.

The anhydrite layers both above and below the salt beds are also chemical precipitates. This is probably also true of the limestones and dolomites associated with the salt.

Table 2 furnishes estimates on depths to salt in the northern part of the state.

AREAL EXTENT AND RESERVES

The area of rock salt deposits in West Virginia and nearby states, shown by wells in which salt is known to be present includes all or a considerable part of Hancock, Brooke, Ohio, Marshall, Wetzel, Tyler, Pleas-

Table 2 . . . Depths to Salt in West Virginia[3]

Locality	Surface Elevation	Estimated Depths to Salt
Chester, Hancock County	700	4,875
Westernmost part of Hancock County, near Ohio River, opposite Yellow Creek	700	4,820
Wellsburg, Brooke County	680	5,940
Wheeling, Ohio County	700	6,355
Greer, Monongalia County	1,460	8,400
Morgantown, Monongalia County	850	9,100
Fairmont, Marion County	900	9,200
Bens Run, Tyler County	640	6,300
Belmont, Pleasants County	640	5,700
Natrium, Marshall County	689	6,724 (actual depth)
2 mi. southwest of Goose Creek, Ritchie County	830	6,523 (actual depth)
Near West Union, Doddridge County	1,138	7,758 (actual depth)
Near Lost Creek, Harrison County	1,113	8,036 (actual depth)

ants, Monongalia, Marion, Harrison, Doddridge and Ritchie counties. The amount of information on thickness of rock salt in West Virginia is meager. While the bed has a thickness of 110 ft in Marshall County, a well in a nearby Ohio County suggests that there may be another bed in West Virginia below the one already penetrated.

Rock salt has a specific gravity of 2.17 and weighs about 135 lb per cu ft. A salt bed with an average thickness of 100 ft, would yield 188 million tons per square mile. Since there are, in West Virginia, at least 2400 square miles underlain by rock salt, it is seen that the total reserves are enormous.

Although its presence has not been shown, there is reason for believing that rock salt may be found in the Silurian rocks of our northeastern counties (Hampshire, Mineral, and Morgan) where the geologic conditions are favorable. This belief is based upon the occurrence of imprints of salt crystals in the Wills Creek and Tonoloway formations and the report years ago of the existence of strong salt springs in the Keyser area.

Salt Production and Use

From the beginning of the salt industry until 1942, all of the salt produced in West Virginia was derived from salt brines, after 1942 production consisted of both brine and rock salt. Fig 1 shows the general trend of salt production in the state as compared with production during the peak year of the early industry in the Kanawha Valley. From 1882 until 1936 the state production never reached the peak that it had obtained in 1846. From 1936 to 1945, however, production grew tremendously reaching a high of 370,260 tons in the latter year with a value of over one million dollars. The early part of this increase from 1936 to 1942, was brought on by the expansion of the chemical industry which depended on salt brine for its raw material. The great increase in production between 1942 and 1945 was made possible by the beginning of the West Virginia rock salt industry and the requirements of World War II.

In the early days salt was produced only for family consumption but as wells were drilled and the salt industry began this broadened out to include production for domestic and agricultural use. Since the beginning of the chemical industry based on salt as a raw material, its uses have multiplied rapidly and now include bleaches, soda ash, dyes, soap, textile processing, packing and curing of foods, dairy products, water treatment, metallurgy, insecticides, synthetic rubber, and others.

Summary and Conclusions

The salt industries of West Virginia have attained large proportions, and will increase further. The reserves of salt both as brine and as rock salt are large. Rock salt reserves under the northern counties are enormous, and their further development along the Ohio River is expected to play an even more important part in the chemical industries of West Virginia.

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Some Aspects of Mechanical Coal Cleaning in Utah

By CARL S. WESTERBERG*

Coal preparation practice and trends follow, among other factors, production trends in any given area. Considering an area the size of a state, some broad predictions may be made after a review of the annual production records. As this paper is concerned with coal cleaning in Utah, data on its production record are pertinent.

Table 1 shows the percentage relation of coal production in Utah to the national total as a single figure up to 1935 which includes all recorded production. From 1935 through 1947 these data are presented by years.

It is evident from these data that the production of Utah coal is ascending. The reason for this trend is industrial growth in the West resulting in a population shift which accounts for more people than the normal increase would supply together with Utah's relatively fortunate geographical location and its abundance of good-quality coal. Utah is also hopeful of supplying more coal in the future for West Coast power generation.[1]

Utah is not unique among coal-producing states in anticipating greater production in the future. *Coal Age* states, " . . . it is now evident that a billion-ton yearly production of anthracite and bituminous by 1960 is not impossible." At present it is about 610,000,000 tons, the all-time high in 1947 was approximately 687,803,000 tons.

In view of the trend outlined above, obviously more new mines will be required to produce the necessary tonnage. These in turn will require more mechanical cleaning capacity. Whether the required new plants will be built by present operators of mines and mechanical cleaning plants or by others now in the business of producing coal or expecting to enter it is of no consequence in this discussion. The fact is—they will be built.

San Francisco Meeting, February 1949.

TP 2632 F. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Aug. 30, 1949. Manuscript received Jan. 28, 1949.

Director of Coal Preparation and Research, Utah Fuel Co., Salt Lake City, Utah.

[1] References are at the end of the paper.

Table 1 . . . Relation of Coal Production in Utah to National Total from 1935 through 1947

Year	Utah Production to National Total, Pct[a]	Utah Coal Mechanically Cleaned, Pct[b]	Utah Production in Thousands of Tons[c]
Total all years records exist to 1935	0.70	0.0	121,600
1935	0.79	0.0	2,947
1936	0.74	0.0	3,247
1937	0.86	0.0	3,810
1938	0.85	0.0	2,947
1939	0.83	9.1	3,285
1940	0.78	20.3	3,576
1941	0.79	22.1	4,077
1942	0.95	23.1	5,517
1943	1.13	18.7	6,666
1944	1.15	22.7	7,119
1945	1.16	23.5	6,679
1946	1.12	26.5	5,990
1947	1.18	24.4	7,429

[a] Calculated by author from U.S. Bureau of Mines data.
[b] Estimated by author. Certain installations are not considered such as Bradford breakers, a spiral chute cleaner, and one small air-sand plant.
[c] U.S. Bureau of Mines.

Other Factors Indicate Need for More Mechanical Cleaning-plant Capacity in Utah

At the present time, two of four metallurgical-coal-producing mines in Utah require that their output be mechanically cleaned. While the exact extent of coking-coal reserves in Utah has not been defined, the minable reserves are not considered to be as extensive as western heavy-industry would like. Therefore, it is necessary to recover all of this type of coal possible with whatever impurities must be taken with it. Thus, efficient iron and nonferrous ore reduction, requiring high-grade coke will need mechanical cleaning plants to produce clean coal from mine-run coal. The latter may contain 30 pct or more of extraneous impurities.

The national policy of conserving natural resources also requires that coal extracted from government-leased land be removed as completely as possible consistent with minability. This does not permit the past practice in private lands of detouring around local areas found to be difficult mining because of poor roof conditions or containing coal with considerable extraneous impurities such as bony bands or rock splits. The U. S. Geological Survey, charged with the administration of public lands, does not demand that a mechanical cleaning plant be erected to process coal from public lands. However, enforceable regulations in mining and completeness of extraction dictated by this agency may make such plants necessary if markets are to be found.

Industrial users of coal are growing more aware of the cost advantages of using mechanically cleaned coal. It is lower in ash and sulphur, moisture content and size consist are uniform, and ash-softening takes place within narrower temperature limits. In addition, the fallacy of paying high freight rates for long hauls for a relatively high percentage of noncombustible material becomes apparent upon analysis. For example, consider the movement of 100,000 tons of coal from the Carbon County, Utah district, to the Bay area of California. The present freight rate is $5.95 per ton plus a 4 cents per ton excise tax. A washed coal with a 2 pct lower ash content would result in a saving of $11,980 in freight on inert material alone. Heat from one typical Utah coal delivered to the Bay area in the raw state would cost 42.6 cents per million Btu. The heat from the same coal washed (2 pct less ash content) would cost 41.6 cents per million Btu. This saving expressed in dollars on a 100,000 ton movement would amount to $24,622. In addition, there are calculable savings to be realized in reduced ash handling with the use of washed coal and tangible but difficult-to-calculate savings from greater uniformity.

Domestic coal users, also prefer mechanically cleaned coal because its use results in trouble-free operation.

Competition with liquid and gas fuel makes it necessary to give the user the most heat and satisfaction for his money. With the trend of rising prices in coal, the producer must increase his prices to stay in business. The customer is obliged to pay the bill of course, but he can keep his cost of heat down by getting better value in purchases of coal. The operator must then, afford a better product through mechanical cleaning.

Thus, the factors that indicate more mechanical coal cleaning will be required in Utah are:

1. More coal will be produced.
2. The supply of coal for metallurgical purposes must be extended by mining and then cleaning the highest feasible portion of this type of coal.
3. The government policy of complete coal recovery from public lands may require mechanical cleaning of the coal mined.
4. Quality requirements by large users is more exacting.
5. Domestic users are more quality-conscious.
6. Competition with other forms of fuel requires that more heat be delivered per unit weight of coal.

Factors to Consider in Planning Mechanical Cleaning of Coal in Utah

The classical engineering approach to any mechanical cleaning problem, after the economics have been studied, is to conduct a washability study. This study, when it accurately represents the coal to be cleaned, gives the following information:

1. The size distribution or screen analysis, necessary to the designer in planning screening, conveying, and drying capacity.
2. The probable ash and sulphur content of the cleaned coal at any given specific gravity.
3. The percentage of coal that can be recovered as clean coal at the washing gravity selected.
4. The difficulty or ease with which the coal can be washed, that is, the probable efficiency of the separation.
5. The type and size of equipment that will be required to perform the necessary operations.

The economics of the proposed plant must then be reviewed in the light of information from this study.

Washing Characteristics of Coals Mechanically Cleaned in Utah

From an examination of the washability curves yielded by such studies shown in Fig 1 and 2 it may be generalized that Utah coals now cleaned mechanically do not present any very difficult problems for the equipment employed. It may also be assumed that mines to be opened in the future will yield coal with characteristics similar to those of one of the four shown.

In each of the coals represented, Curve *A* indicates the percentage of coal that will float at a given specific gravity. The ash and sulphur content (not shown) is generally plotted on an extension of this graph. By placing a straightedge across the graph, the percentage of recovery and ash and sulphur content of both the clean coal and rejects can be determined at any given specific gravity of separation.

Curve *B* on each graph is a convenient way to obtain some idea of how difficult the washing problem will be. The position of this curve is derived from Curve *A* and shows the amount of material in percentage included within 0.05 sp gr on either size (plus or minus) of the separating gravity. In each of the washability curves illustrated the corresponding coal is washed at approximately 1.50 sp gr. Table 2 shows what may be inferred from these graphs.

Table 2 . . . Washing Characteristics of Coals at 1.50 Sp Gr as Indicated in Fig 1 and 2

Coal	Indicated Recovery, Pct	0.05 Sp Gr	Ease of Separation
King mine....	93	1.5	Very easy
Clear Creek...	93	1.5	Very easy
Castle Gate...	95	4.0	Moderately difficult
Sunnyside.....	94	3.0	Easy

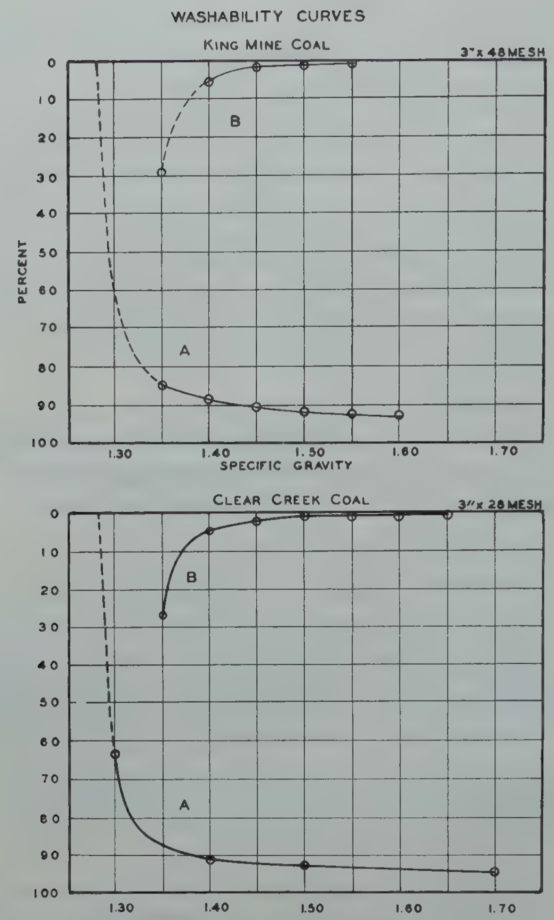

FIG 1—Washability curves. *A*, specific gravity curve; *B*, ± 0.05 sp gr distribution curve.

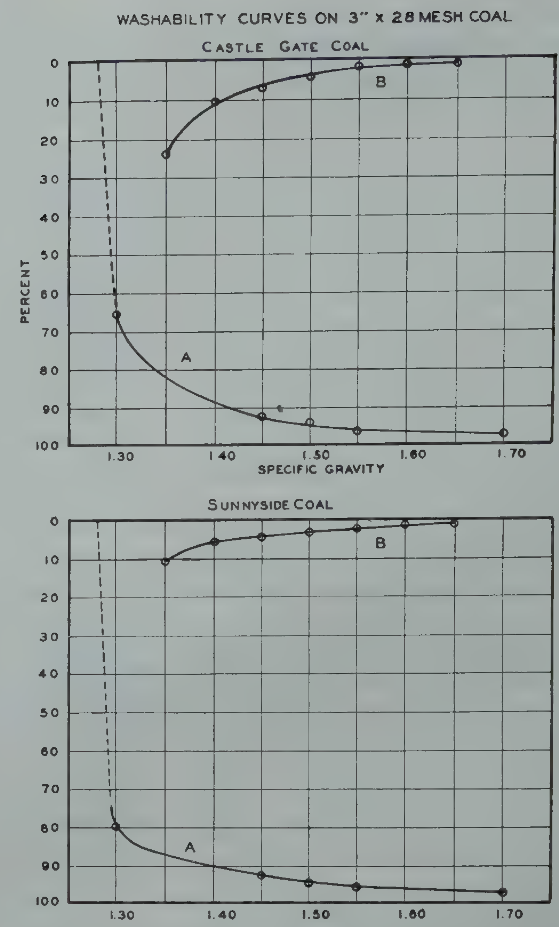

FIG 2—Washability curves on 3 in. by 28 mesh coal. *A*, specific gravity curve; *B*, ± 0.05 sp gr distribution curve.

The author has seen charts which assign descriptive terms to arbitrary numerical limits on the 0.05 sp gr curve. These are informative, but are not shown here because they are devised for eastern coals which are somewhat different than those illustrated and consequently may not apply. In other words, the terms "very easy," "easy," etc., as shown on Table 2 are relative.

These four coals are washed in three plants equipped with Baum-type jigs. All have five cells with two compartments and are pulsated by air at from 2 to 3 psi pressure. The rated capacity of each is about 250 tph and all are operated in excess of rating with results consistent with the varying character of the coals and the rate and continuity of feed.

Two of the plants are for the preparation of domestic and industrial fuel and the third is for metallurgical coal. Centrifugal drying is employed in all three plants on sizes below $\frac{3}{16}$ in. and in the two domestic and industrial coal plants, heat drying for the sizes below 1 in. is also used.

Fig 3 presents the approximate performance of these plants with respect to product quality and a comparison with the raw coal fed to the plant.

It will be noted that the limits of ash and sulphur content on the washed products are not wide, but better performance in this respect would be desirable. Unfortunately, extremely close control of the ash and sulphur content of the washed coal output cannot be realized without a uniform input. To ensure a uniform raw coal feed to a cleaning plant requires a large and suitable blending bin. It should have a capacity related to the daily average production of the mine with a minimum of about 800 tons of storage space to be effective. This bin must be compartmented and equipped with an arrangement to draw-off coal from all of the pockets at the same time. Such special bins have fairly wide acceptance in eastern metallurgical-coal-cleaning plants but are not generally used in cleaning plants processing domestic and industrial coal. However, this facility can be justified from another viewpoint.

A large surge capacity for raw coal makes the mining operation more efficient. With a surge-blending bin, moderate-size cleaning plants can prepare a constant but smaller flow of coal as compared with an oversize plant necessary to accept an intermittent flow with high peaks in the rate of input.

Thus by the use of surge-blending bins high-cost mining equipment may be used continuously and more profitably without the necessity of building more expensive additional cleaning capacity.

The Geneva Steel Co. in Utah employs such a blending bin though not in connection with a cleaning plant. This firm has evidently planned its operation on the assumption that it is better to have a uniform raw coal for its coking operations than a cleaning plant which may not be necessary at

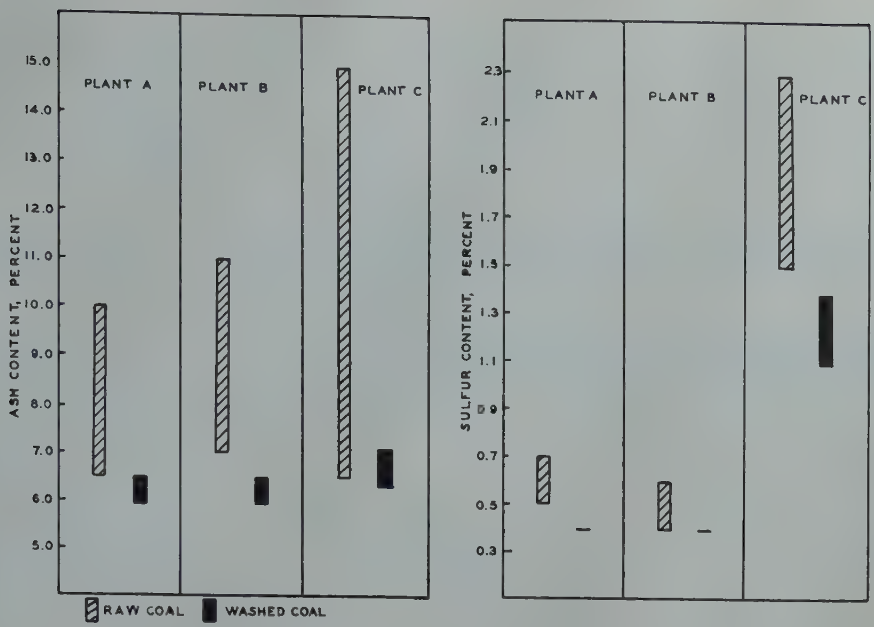

FIG 3—Approximate performance of three mechanical cleaning plants on representative Utah coals.

the present time. At any rate, it has the basic step completed when it is ready to employ mechanical cleaning. Meanwhile, this company is realizing the advantages gained in continuous mine operation.

A Cleaning Plant Byproduct

An interesting feature of the mechanical cleaning plant operated by the United States Fuel Co., at Hiawatha, Utah, is the recovery of fossil resin which occurs in the coal. This material, when pure, is useful in the paint and varnish industry. When the Combined Metals Reduction Co. perfected an economic method for concentrating free resin from coal,[2] an alert management at the Hiawatha property added facilities to skim off impure but concentrated resin from the circulating water circuit in the cleaning plant. This concentrate is sold to Combined Metals Reduction Co. for further processing. By this means a nuisance material is converted to a valuable byproduct. This free resin does not occur in sufficient quantities in coals mechanically cleaned in other Utah plants to warrant recovery.

Baum Jig-type Plants

Any western operator, considering the installations of mechanical cleaning facilities should study these existing plants because they are available, operate satisfactorily, and represent considerable thought, planning, and experience. Because water is required for operation, the operator should examine his own supply to be sure he can rely on at least 300 gpm during operating time. If, as is often the case in a semiarid location, water in this quantity is not available, he must turn to other processes.

Other Systems of Mechanical Coal Cleaning

Coal may be air cleaned by the use of air and air-sand combination, between the size ranges of from about 4 in. down to 48 mesh. However, thought must be given to adequately preparing the coal prior to cleaning. Recent eastern installations have shown that air systems are only as successful as are the drying and screening equipment ahead of the cleaning units. It is not overcautious to state that it is necessary to heat dry *all* coal before screening and air cleaning if reliable performance is expected.

Where precision washing is required, other types of washing equipment merit consideration. Recent perfection of heavy-density cleaning equipment permits very precise control of the washed product. In this system, heavy foreign matter is separated by gravity from coal by the use of a medium consisting of water and magnetite, sand, or other materials which have the effect of increasing the specific gravity of the water to any desired specific gravity from about 1.3 to 1.7. This system is inherently more precise than a system that depends upon stratification of material in a rising current of water as in the Baum jig, hydro-classifier, or launder. In fact, the heavy-density system yields almost a theoretical separation. There are two major objections to such a system, it will not clean coal much below ¼ in. by 0 in size, and the cost of installation and makeup media is relatively high. In the case of the sand system, maintenance is relatively costly. However, the system is justified in specific cases.

Recovery of Coal

The washability study, when gravity separations are made down to zero size, yields a percentage of possible recovery based on the idea that all the coal down to zero size, floating at a given specific gravity, is recoverable. In practice, the approach to this ultimate recovery is only as close as the equipment provided will allow. Coal below minus 28 mesh in size becomes increasingly harder, that is, more costly to recover. Until about a year ago it was not possible to recover clean coal much below 48 mesh in size because the equipment to do so

did not exist. This unrecoverable clean coal represented from about 2 to 5 pct of the raw feed to a preparation plant. The accumulation of vast amounts of "slurry" in nearby ponds became a costly nuisance, and in addition, represented unrecovered dollars. Various devices have been used to reduce this loss, none of them a real solution to the problem.

Because of handling difficulties it is impossible, or prohibitively expensive, to recover this slurry as such by heat drying alone. Some operators found it possible to bypass some portion of the $\frac{1}{4}$ in. by 0 raw feed and remix it with the washed portions of coal. This did reduce the loss of fine coal, but close control of quality in the washed product was lost.

Now available is a continuous solid-bowl centrifuge which appears to be able to dewater coal down to almost zero in size. Application of this device may solve the fine-coal-loss problem in wet cleaning plants. The Marianna Plant of the Bethlehem Steel Co. employs several units of this type and is said to operate in completely "closed circuit," that is, it has no loss of fine clean coal. Other plants now under construction will employ these units.

The Utah Fuel Co. has completed a preliminary study of the cyclone thickener[3] as a possible partial solution to this problem. Application of this device in connection with heat drying, it is hoped, will help to solve the fine coal recovery problem in its Utah preparation plants. It has been tried elsewhere with doubtful success, but it is believed that further refinements and a better understanding of its application will make it a valuable tool in fine coal recovery.

Another factor to be weighed carefully is how far to go in the matter of heat drying. In preparing domestic and industrial coal, it has been sufficient to remove enough surface moisture to ensure that the stoker sizes do not freeze during the winter and that this size handles properly in automatic equipment. To do this, heat drying is necessary after taking full advantage of mechanical dewatering equipment. In long-range planning it is advisable to provide space for additional heat drying equipment for the coal sizes below 1 in. because expanding Utah coal markets will probably include a substantial tonnage movement over long distances. In this case, a producer with extra drying capacity will have a competitive advantage.

Detailed Planning Necessary

Careful attention to details will yield large dividends in efficient plant operation when a new plant begins operation. Some of these are:

1. Provision for adequate equipment to move refuse and a space to dispose of it.
2. A good system for oil-treating domestic stoker coal.
3. A good heating system for the preparation plant.
4. An adequate supply of repair parts in stock and readily available.
5. Lockers for special tools and equipment within the preparation plant.
6. Plenty of artificial lighting within the plant and in the railroad car-loading and storage area.
7. A good control laboratory supervised by a trained technician with sufficient help to carry out an adequate control program.
8. Provision for inspection maintenance at regular periods every operating day.
9. Conveniently located power, compressed air and welding current outlets within the plant.
10. A good communication system between control men and various points in the plant and the railroad car-loading space.
11. Making all portions of the plant easily accessible for inspection and maintenance.

Costs of Mechanical Cleaning

In 1938 and 1939 a plant similar to those erected in Utah in that period cost approximately $1000 per ton-hour of capacity. In 1942 and 1943 another similar Utah plant cost double this amount. At the present time a cleaning plant would cost about $4000 per ton-hour of capacity. This cost is high and prohibitive for smaller operations. It is interesting to speculate on what could be done by organizing a cleaning plant company to serve a group of moderate and small sized operations in the same district.

Summary

The foregoing presents some of the reasons why coal mechanical cleaning capacity in Utah may be expected to increase. To meet the need for increased capacity for coal cleaning in Utah, one approach to the selection of equipment is outlined with mention of some of the special problems that may be encountered.

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Symposium on Western Phosphate Mining

CONTENTS

Foreword

By E. M. NORRIS,* Member AIME

Phosphate deposits are distributed widely over the earth's surface. Of the known areas of deposit, eight fields are of particular interest because of their vast reserves of high grade phosphatic material and the volume of their production that goes to the world's great consuming areas. These fields, in the order of their present economic importance, are situated respectively in Florida, French Morocco, Kola Peninsula of U.S.S.R., Tunisia and Algeria, Tennessee, Montana-Idaho-Wyoming-Utah, Ocean and Naru Islands in the Gilbert group, and in Egypt.

The Rocky Mountain deposits have a comparatively low ranking in the phosphate trade of today, but this field contains one of the largest known reserves of phosphatic material, associated with which are recoverable trace minerals, which are of increasing importance in our national economy. The first discoveries of phosphate rock in the western states were recorded in Cache[1] and Rich[2] counties in Utah, respectively in 1889 and 1897. Mining operations were begun on these deposits near Montpelier, Idaho, in 1906. During the latter year geological studies of the phosphate beds were started by Weeks and Ferrier of the U. S. Geological Survey. These studies have been continued to the present day by a distinguished roster of the Survey's technicians. As a result of this systematic exploratory program, the Survey has published a series of bulletins, papers, and maps, which provide the mining public with a comprehensive study of the exposed

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[1] References are at the end of this paper.

portions of the deposits. These publications have been of great value to those engaged in the commercial development of this field. To the hoary western miner, it is indeed a glimpse of Utopia, when sourdough and geologist, alike, have the well-thumbed pages of "Mansfield"[3] to guide their questing footsteps.

Production from the western field lagged for many years because of its remoteness from fertilizer markets and the resulting heavy transportation costs. During the years of 1919 and 1920, pilot plants were put into operation at Anaconda, Mont., for the manufacture respectively of sulphuric acid and high-analysis acid phosphate.

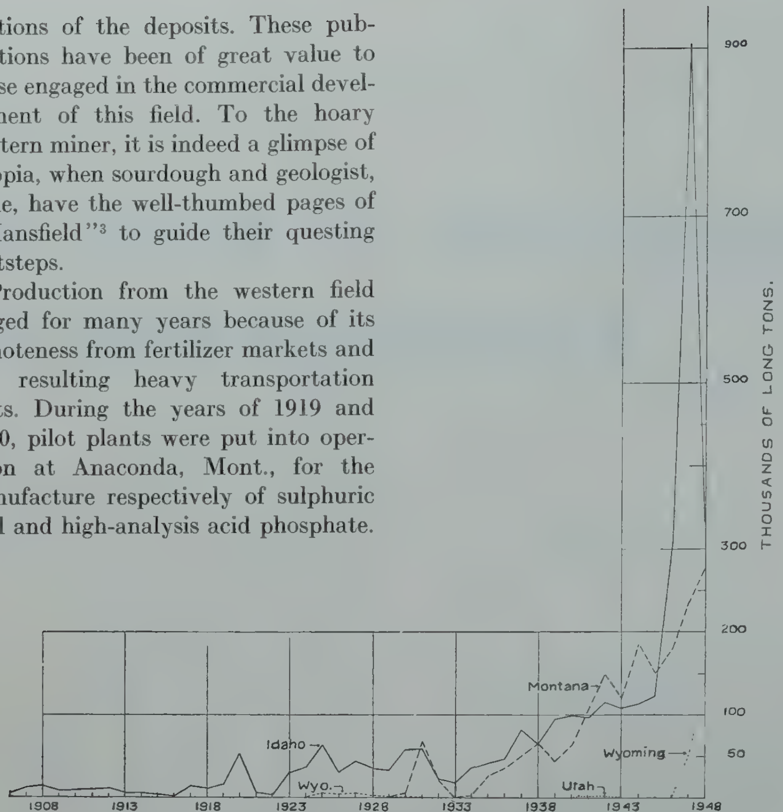

FIG. 1—Idaho, Montana, Wyoming and Utah phosphate rock production, 1906 to 1948.

These plants were soon replaced by full scale operations. Fourteen years later enterprising agronomists of the western beet-sugar manufacturers discovered that proper application of phosphate fertilizers would increase sugar-beet yields, in the alkaline soils of the intermountain states, by approximately 16 pct. Thus a western fertilizer market was created and the advent of World War II caused a rapid expansion in production from the western deposits (Fig 1). Four major producers are now operating in this field and a number of properties are in various stages of development. It will be the purpose of this symposium to describe the most recent geological and mining developments.

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Geological Studies of the Western Phosphate Field*

By V. E. McKELVEY†

Introduction

The Phosphoria formation of the northwestern states presents a stimulating challenge to workers in every field of mineral technology. In addition to its large reserves of phosphate, the formation has been found to contain important amounts of fluorine and several metals, such as vanadium, nickel, and molybdenum. Because these minor metals in the Phosphoria do not have their usual glitter, concentrations of them are not easily discovered nor appraised. Like the phosphate itself, production of these metals very likely can be achieved only through use of costly chemical methods, for their mineralogy precludes treatment by the usual methods. In addition, in much of the area the containing rocks lie in complexly folded and faulted structures, which complicate mining and increase its cost. Yet the chain of research necessary to establish what elements are present; their geographic and stratigraphic distribution and reserves; the most suitable methods for recovering them; the combination of beds which, when mined and processed, will yield the greatest recovery of the most valuable elements; and the methods most adaptable to their profitable extraction may well lead to the development of an important mineral industry and may make possible a significant reduction in the cost of fertilizer—this in itself would be an achievement of momentous importance—and at the same time make the nation more self-sustaining in critical materials.

Although this research has great

* Published by permission of the Director, U. S. Geological Survey. Manuscript received Dec. 13, 1948.

† U. S. Geological Survey, Spokane, Wash.

economic promise, most of the problems have proved so large, complex, and costly that a single company cannot afford to attack them, though all the larger companies have made important studies of certain technologic problems of immediate interest to them. Considering possible benefits to the nation, however, the odds against success are not too great for the public to risk, and it is therefore highly appropriate for the Federal Government to investigate the several types of problems named.

The U. S. Geological Survey's part in this investigation, in which a number of other agencies are participating, is to define what elements are present, where they occur, how much of each is present, in what structures the containing beds lie, and the origin of the rocks, and the elements contained in them. To achieve these various purposes satisfactorily we are examining previously unmapped areas in which the Phosphoria formation could occur, but has not yet been looked for or found; and we have begun to map, on a scale no smaller than 1:62,500, the geology and topography of all areas containing the Phosphoria formation; to measure, describe, and sample all beds of the phosphatic and shaly portion of the formation at scores of localities over the western field; and to analyze the samples for phosphate, fluorine, vanadium, minor metals, and, for certain beds, rock-forming constituents.

The mapping and sampling will (1) define the regional geologic structure, and (2) permit estimation of reserves (of phosphate, fluorine, vanadium, and other elements of potential value) of the "inferred" class over the entire region and, in certain areas, reserves of the "indicated" class as well. These data should thus provide sufficient basis for industry to make preliminary selection of sites and of combinations of beds suitable for mining, but the detailed sampling necessary to prove (or measure) reserves and the drilling and large-scale mapping (i.e. 50 to 100 ft to the inch) necessary to work out details of structure in the degree necessary to plan actual mining operations will have to be done by the companies themselves. We feel this is the practical as well as the ideal division of effort between federal and private geological studies, for though the federal investigations outline the regional picture in a detail which facilitates and stimulates the work of private industry, it leaves to industry those studies incident to actual production.

Though it is not possible to give in this paper a full account of even new information acquired on the geology of the western field, some of the more salient features of the regional geology and the phosphate deposits are discussed in the following pages. In addition, the occurrence of one of the other trace constituents—uranium—is discussed briefly in another paper.[1]

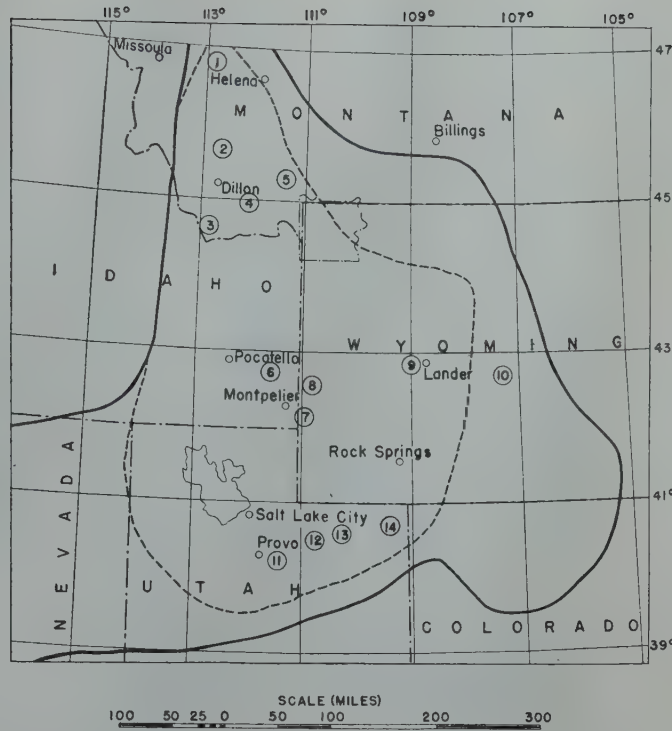


FIG 1—Index map of the western phosphate field, showing limits of the Phosphoria, Park City, and Embar formations (solid line) and their phosphate deposits (dashed line).

The eastern and southeastern boundaries shown on the map fairly accurately represent the true limits at the time of deposition. The western limits at the time of deposition cannot now be accurately reconstructed because the Permian rocks there either have been eroded away or are concealed beneath thrust plates of older rocks. The southern and southwestern limits are now poorly defined because not enough stratigraphic work has been done to differentiate the facies of the Permian Phosphoria formation from those of the Kaibab limestone, Gerster formation, and Arcturus formation and other western facies which may be at least partial age equivalents. Numbers show the location of measured sections on Fig 3, 4 and 5.

Geologic Setting of the Western Phosphate Field

Phosphate deposits in the Permian Phosphoria formation and its partial stratigraphic equivalent in Utah, the Park City formation, crop out in an area of some 100,000 square miles in Montana, Idaho, Wyoming, Utah, and Nevada[a] (Fig 1). This area may be divided into an eastern portion of relatively simple geologic structure and a western portion of complex geologic structure. The boundary between the two provinces is irregular but roughly approximates meridian 111° (Fig 2).[a]

In the eastern part of the field the dips are gentle and the folds are the open-dome and basin type which have no dominant orientation. Thus the Uinta Mountains of Utah and the Centennial Range of Montana strike east, the Wind River Mountains and the Gros Ventre Range of Wyoming strike northwest, and the Rock Springs anticline of Wyoming strikes north. Though numerous faults, including some thrusts, are found, large unfaulted areas are present also. Eastward, near and beyond the margin of the Phosphoria formation, the folds become less numerous, even less complex, and gradually die out toward the plains.

[1] References are at the end of this paper.
[a] Discovered recently in the Goshute Range by H. E. Wheeler, personal communication.

[a] Details of the regional structure can be seen on the many published quadrangle maps over the field; the general picture, however, is best shown on the Tectonic Map of the United States: Amer. Assn. Petr. Geol., 1944, and the Geologic Map of the United States: U. S. Geol. Survey, 1932.

Range structural province.

The structure of the western part differs somewhat from east to west across the field. Near the eastern margin, as in the Wasatch Mountains of Utah, the Salt River Range of Wyoming, and the Tendoy Mountains of Montana, most of the folds are overturned, many of them isoclinal and of very short wavelength; the mountains or the folded belts are tens of miles in width and are identifiable for scores of miles along the strike, whereas the structural valleys are narrow or of the order of only a few miles in maximum width. In addition, the rocks cropping out at the surface are predominantly of late Paleozoic and Mesozoic age. Farther west, on the other hand, the folds are more open, fewer of them are overturned and their wavelength is longer; the folded belts or mountain ranges are less continuous along the strike and in much of the region (south-central Idaho, western Utah, and eastern Nevada) the basins are wider and occupy a larger proportion of the region than to the east. Although late Paleozoic and Mesozoic rocks are found, the proportion of pre-Carboniferous rocks cropping out at the surface is much higher than to the east. The structure is complex, however, because of the abundance of large overthrust faults, and, in fact, it is these overthrusts that are in large part responsible for the numerous outcrops of the pre-Carboniferous rocks.

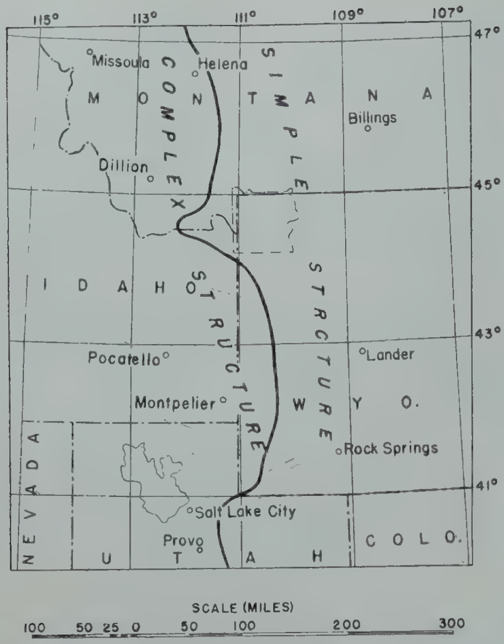


FIG 2—Boundary between areas of simple and complex geologic structure in the northern Rocky Mountains.

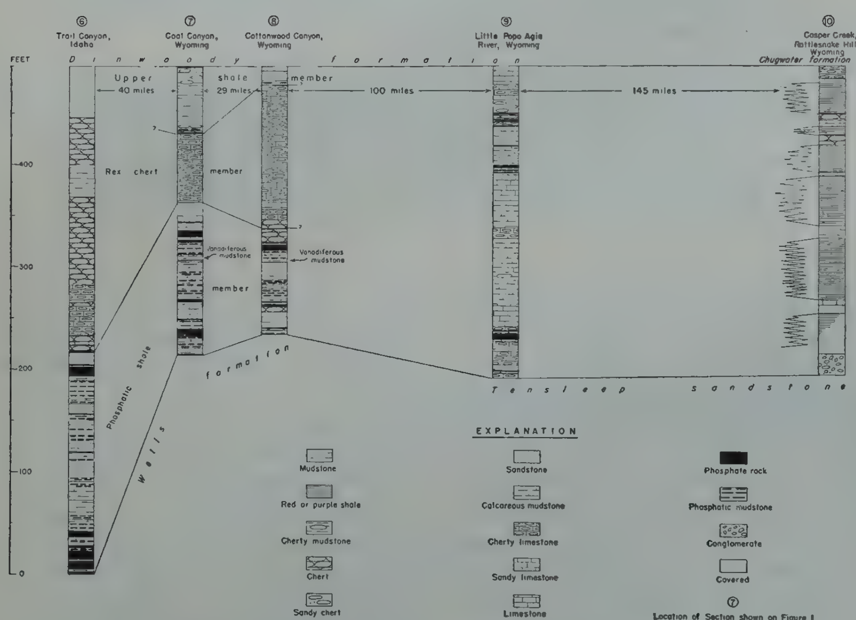


FIG 3—Typical sections of the Phosphoria formation in Wyoming and Idaho.

The Rattlesnake Hills section was measured by H. D. Thomas;[4] the Lander section by Ralph H. King;[11] the Salt River Range section by J. D. Love and L. E. Smith; the Coal Canyon section by V. E. McKelvey, and the Trail Canyon section by L. E. Smith, R. A. Hoppin and V. E. McKelvey, all of the U. S. Geological Survey.

In marked contrast, the western portion of the field is characterized by steep dips and tight, closely spaced folds, many of which are overturned. Overthrust, reverse, and transverse faults of both large and small displacement are abundant—in fact it is unusual to find outcrop segments of more than a mile in length which are not broken by faults, and it is not uncommon to find outcrops of the Phosphoria formation so faulted and mashed as to be altogether unsuitable for mining. Normal or valley faults of large displacement and relatively recent origin are also abundant in the western portion of the field, and, though it is not generally so classed in textbooks, this part of the region is in reality a part of the Basin and

Regional Variations in the Phosphoria Formation

The Phosphoria formation is noted for the great lateral continuity of the beds composing it, but it displays significant lithologic variations over the field as a whole. At its type locality in southeastern Idaho,[2] the Phosphoria formation consists of a lower, phosphatic shale member about 180 ft thick and an upper member, the Rex chert member, 240 ft thick; another member, a cherty mudstone or shale 15 to 75 ft thick overlies the Rex member in most of southeastern Idaho, though it is not well-defined at the type locality of the Rex member in the Crawford Mountains of Utah.[3] These units of the Phosphoria are easily recognizable over a wide area in Idaho and adjoining parts of Wyoming and Utah, but in central Wyoming the whole aspect of the formation is different, for it is thinner, and contains a greater pro-

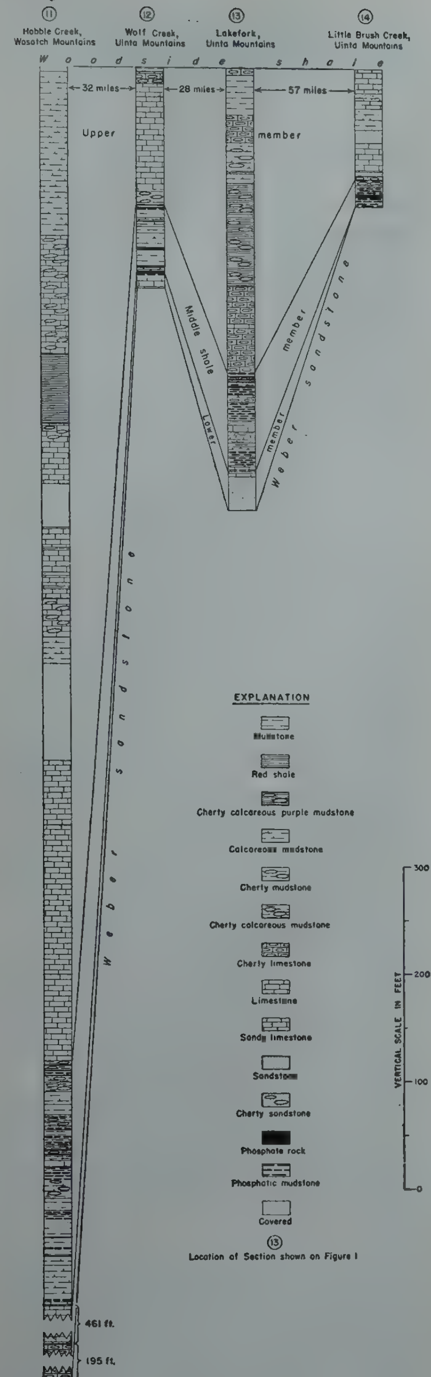


FIG 4—Typical sections of the Park City formation in Utah.

The Little Brush Creek section was measured by D. M. Kinney; the Lake Fork, and Wolf Creek sections by J. W. Huddle; and the Hobble Creek section by A. A. Baker, R. S. Sears, M. D. Stewart, G. F. Hosford, and D. P. Sprouse, all of the U. S. Geological Survey.

Note: the line at the top of the Park City formation may not be the same time-line at every locality, for the upper part of the Park City formation tongues out into red beds similar to the Woodside shales in the eastern part of the area; see H. D. Thomas and M. L. Krueger.[8]

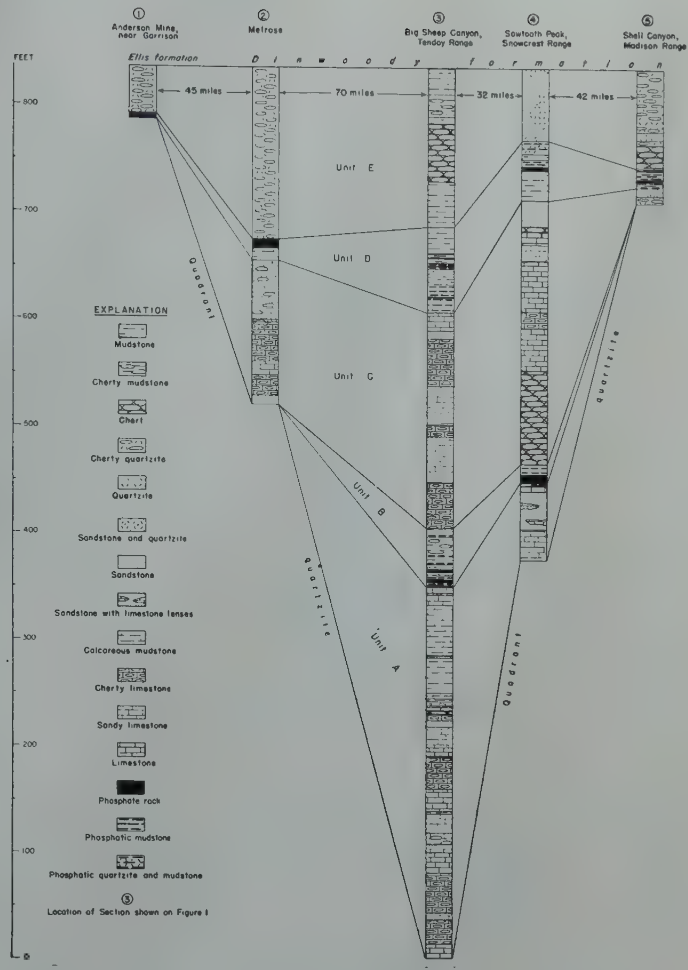


FIG 5—Typical sections of the Phosphoria formation in Montana.

The Madison Range section was measured by R. A. Swanson; Sawtooth Peak section by F. S. Honkala and O. A. Payne; Dell section by W. R. Lowell; and the Melrose and Garrison sections by M. R. Klepper, all of the U. S. Geological Survey. Stratigraphic correlation by M. R. Klepper.

portion of sand and carbonate and much less phosphate and shale. Farther east, in southeastern Wyoming, the phosphate is entirely absent and the formation tongues out into nonmarine red beds[4] (Fig 3). Although the better-known phosphate deposits lie east of Fort Hall in Idaho, the Phosphoria formation has been identified farther west near Malta, Idaho.[5] Both the lower phosphatic shale member and the Rex chert member are reported to be present there and to total about 900 ft in thickness.[6] The quality and thickness of the phosphate beds, however, are unknown.

The westward thickening of the Phosphoria formation is displayed in the other states and so, to some extent, are the other lithologic variations. In Utah the stratigraphic unit known as the Park City formation is the partial equivalent of the Phosphoria, and at Park City, its type locality,[7] it is about 590 ft thick and consists of a lower limestone member, which may be stratigraphically equivalent to the upper part of the Wells formation in southeastern Idaho; a middle shale member (phosphatic, but containing no high-grade phosphate beds) equivalent to the phosphatic shale; and an upper limestone member, equivalent to the Rex chert member of the Phosphoria formation of southeastern Idaho. Eastward the lower member thins out, the phosphate deposits disappear, and the shale and upper limestone members are more clastic and finally tongue out into nonmarine red beds in eastern Utah and western Colorado[8,9] (Fig 4). As in Idaho, the Park City formation thickens markedly to the west and contains a greater thickness of chemical precipitates. A section recently measured by Newell[10] in the Confusion Range near the western border of Utah is 4500 ft thick, though its phosphate content is unknown. In southwestern Montana new investigations by M. R. Klepper, W. R. Lowell, A. P. Butler and other geologists of the Geological Survey show that the Phosphoria formation consists of from two to five lithologic units (Fig 5), provisionally termed

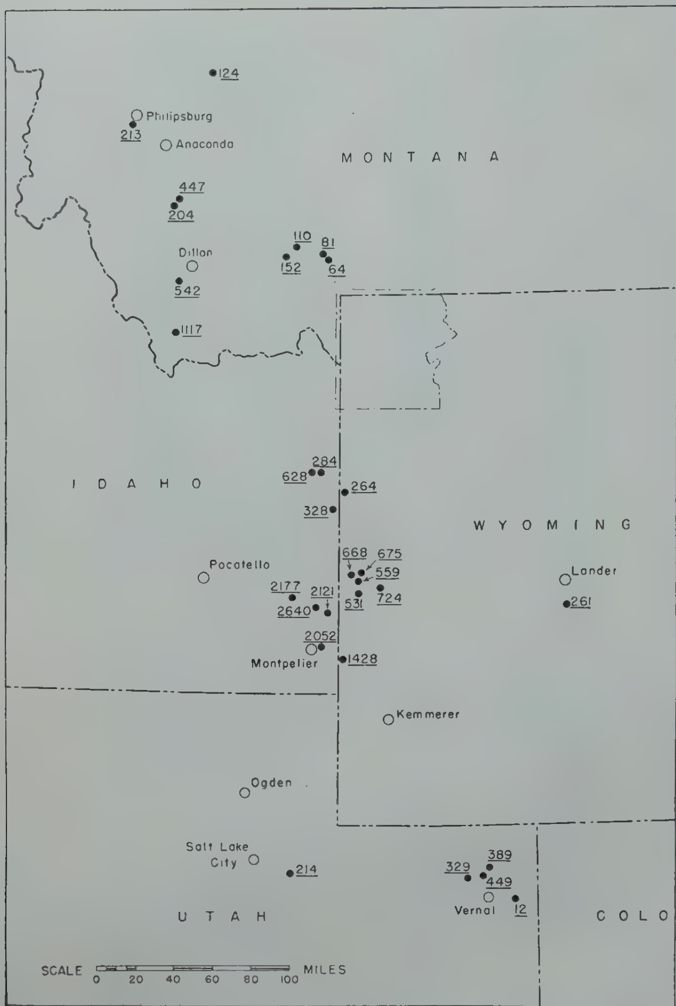


FIG 6—Total phosphate (in feet times percentage of P_2O_5) in the phosphatic portion of the Phosphoria and Park City formations.

units *A*, *B*, *C*, *D*, and *E*. Unit *A*, at the base, consists of a sequence of cherty carbonate and clastic rocks which may prove to be the equivalent of the upper part of the Wells formation in southeastern Idaho, and the lower limestone member of the Park City formation of Utah. Unit *B* is composed of phosphate rock and phosphatic mudstone, unit *C* consists mainly of carbonate rock, unit *D* of phosphatic mudstone, and unit *E* of chert. In the Centennial Range the upper chert, unit *E*, is overlain by a thin section of dark-colored mudstone, which has not been recognized elsewhere in Montana. In the vicinity of Dell and Lima the total thickness of the Phosphoria is 485 ft. The clastic-carbonate sequence (unit *A*), the lower phosphate (unit *B*), and the lower carbonate (unit *C*) disappear eastward in the Madison Range and northward in the Garrison-Drummond area.

In summary, the Phosphoria formation along the eastern margin of the marine basin in which it was deposited contains no phosphatic rocks and consists of thin marine layers, principally carbonate rock and sandstone, interbedded with nonmarine red beds. Westward, as in central Wyoming and eastern Utah, thin phosphatic rocks are interbedded with limestone, mudstone, and sandstone; still farther west, as in western Utah and southeastern Idaho, the formation thickens, and the bulk of it is composed of chemical precipitates (limestone and phosphate rock), very fine detritus (clay and silt), and organic matter. In Idaho the thickening of the formation is in a large part a result of the increased thickening of the phosphatic rocks; in Utah and Montana it is due largely to increase in the amount of fine detritus and carbonate and, though the amount of phosphate is greater too, it is diluted by the other sediments; the phosphatic rocks are not as high in quality, therefore, as they are in Idaho.

Regional Variations in the Thickness and Quality of the Phosphate Rock

In the Bear River Region of southeastern Idaho and adjoining parts of Utah and Wyoming, nearly all the phosphate in the Phosphoria formation is concentrated in two zones—a lower and an upper zone—of the phosphatic shale member, and the thickest and highest-quality phosphate beds are at the base of the lower zone and the top of the upper zone. In southeastern Idaho, the lower phosphate bed is the only one mined over most of the region. The upper bed has been mined intermittently at the Conda mine of the Anaconda Copper Mining Co. In Wyoming and Utah, on the other hand, the lower phosphate bed either is not present or is not as phosphatic as it is in southeastern Idaho, but the upper phosphate bed is thicker and more phosphatic; consequently the upper phosphate bed is the only one mined at most places in western Wyoming and northern Utah.

The middle shale member of the Park City formation, which is the equivalent of the phosphatic member of the Phosphoria in southeastern Idaho, also contains the principal phosphatic beds in Utah; none is as thick and as highly phosphatic, however, as in the Bear River region. In the central Wyoming area, near Lander, two thin, moderately phosphatic zones are present, one near the base of the formation and one about 100 ft below the top.[11] Neither zone is mined at present.

In Montana, phosphate is found at two principal horizons, one near the base of the Phosphoria formation, and another near the middle of the formation. As far as known, the lower zone is of commercial interest only in the Snowcrest and Centennial Ranges, where it is 4 to 6 ft thick, and contains about 32 pct P_2O_5. Elsewhere it is too thin (as in the vicinity of Dillon), or too low in phosphate content (as in the vicinity of Dell) to be mined at present, or is absent altogether, as in the Madison Range, and the Melrose, Phillipsburg, and Garrison areas. The upper phosphate bed is about 4 ft thick and contains about 32 pct P_2O_5 in the Garrison area (where, because the lower members are absent, it occurs at the base of the formation), but farther south in the Melrose, Dillon, and Dell areas the phosphate is so diluted by interbedded and admixed mudstone,

that no thick, highly phosphatic beds are present. The upper phosphate zone is present in the Madison and Centennial Ranges, but is too thin and too low in phosphate content to be mined.

The general westward increase of phosphate content is shown by a comparison of the total amount of phosphate, in terms of thickness multiplied by percentage of P_2O_5, at various localities over the region. Thus, in the vicinity of Lander, the total feet-percent of phosphate in the phosphatic portions of the formation is about 260, and the total increases progressively westward to about 600 ft-pct in western Wyoming, 1400 on the Wyoming-Idaho border, and 2000 to 2600 in southeastern Idaho (Fig 6). Though a similar trend exists in Utah and Montana, at no place yet known does the formation contain as much total phosphate as it does in southeastern Idaho.

Total phosphate alone, of course, is not a reliable means of evaluating the relative quality of the deposits over the region, for a large total phosphate content can be dispersed over a large thickness of rock too low in P_2O_5 content to be minable, as it is in parts of southwestern Montana. Conversely, a relatively small total amount of phosphate, say 160 ft-pct, can be concentrated in one minable layer, as it is in the Garrison area and Centennial Range, where virtually all the phosphate in the formation is concentrated in one minable bed belonging to the middle and lower zones, respectively. Over the remainder of the field, however, there is good correlation between the total amount of phosphate and the thickness of high-grade beds. Thus, the Lander area contains no high-grade beds at all, and the grade increases westward to southeastern Idaho where 11 to 19 ft of beds contain 31 pct or more P_2O_5 (Fig 7). So it is too with beds of lower phosphate content; in the Lander area about 3 ft of beds contain over 25 pct P_2O_5 and in southeastern Idaho 20 to 32 ft of beds contain more than 25 pct P_2O_5 (Fig 8). Similarly in the Lander area only about 4 ft of beds contain more than 18 pct P_2O_5, whereas in southeastern Idaho 43 to 62 ft of the section contains more than that amount (Fig 9).

The conclusion that the Bear River region contains the most and best phosphate is inescapable, but the fact that significant production is coming from Montana now, and that large phosphate reserves exist there and in other parts of the field should not be overlooked. One high-quality phosphate bed only 4 ft thick in a favorable structural and geographic situation can be mined, even though there may be no other phosphatic beds in the section. Regional trends in phosphate content are therefore not to be relied on too heavily in selecting potential mine sites. Nothing short of an exposed and sampled section will prove conclusively that a minable bed is not present at any given locality over a large part of the field.

Though regional variations in the stratigraphy must be considered in an appraisal of the potentialities of the field, they should not be emphasized so as to overshadow one of the most important features of the Phosphoria formation, namely, the remarkable lateral continuity of individual layers in particular areas. Thus, a distinctive phosphate bed at the contact between the Phosphoria formation and the underlying Wells formation in the Bear River region is identifiable from the Crawford Mountains of Utah to Fort Hall, Idaho, a distance of more than 100 miles, although the bed is never over a few inches thick. The lower phosphate bed is mined at various localities between Montpelier and Fort Hall, and over this distance it is relatively uniform in thickness and lithology. Many similar examples of great lateral continuity of individual layers could be given. The lateral continuity of the beds is not great in the eastern and northern parts of the field, but even there the lateral continuity in most places is sufficient to greatly simplify evaluation of grade and tonnage of any particular layer. In contrast with most other types of mineral deposits, as few as 8 or 10 carefully cut samples are generally adequate to evaluate the phosphate content of any particular bed along an outcrop as much as a mile or two in length.

Early reserve estimates[12–15] were based upon reconnaissance data and calculated in such a way as to show

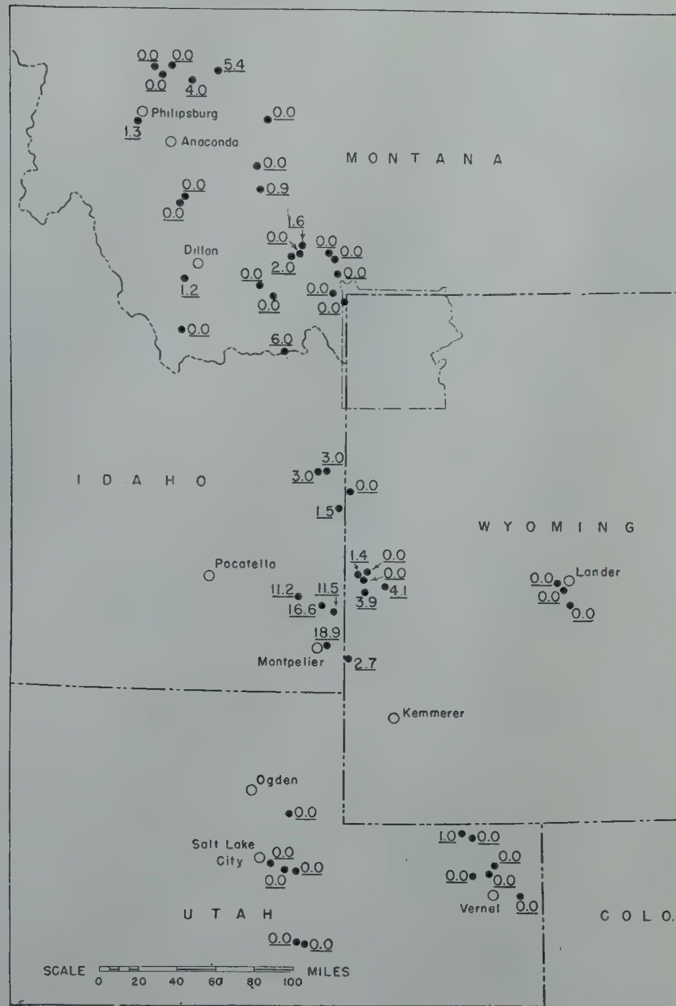


FIG 7—Total thickness (in feet) of rocks in the Phosphoria and Park City formations containing more than 31 pct P_2O_5.

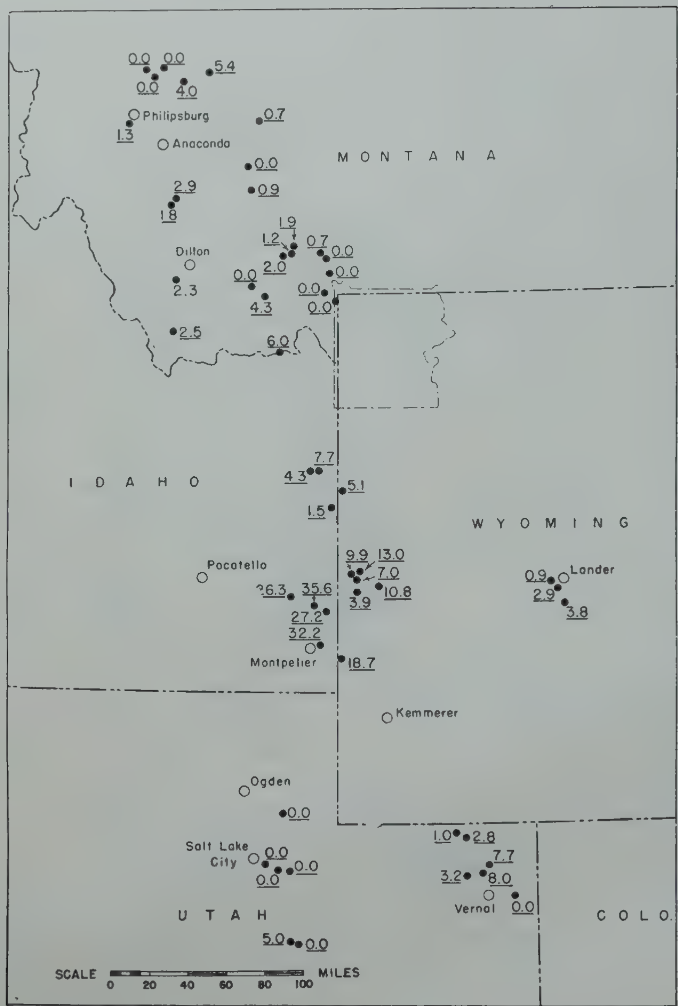


FIG 8—Total thickness (in feet) of rocks in the Phosphoria and Park City formations containing more than 25 pct P_2O_5.

Table 1 . . . Preliminary and Incomplete Estimate of Reserves of Phosphate Rock Suitable for Mining in Permian Rocks of the Western Field[a]

(Millions of Short Tons)

Area	Suitable for Open-cut Mining		Suitable Only for Underground Mining (Above Entry Level)	
	+31 Pct P_2O_5	24–31 Pct P_2O_5	+31 Pct P_2O_5	24–31 Pct P_2O_5
Montana	Not estimated[b]		50[c]	75[d]
Teton Basin, Wyoming and Idaho	None	Not estimated	Not estimated[e]	
Western Wyoming	Not estimated		Not estimated[f]	
Bear River region, Idaho, Wyoming and Utah	25	75	300	1,000
Lander area, Wyoming	None	Not estimated	None	25[g]
Central Wyoming, excluding Lander area	None	Not estimated	None	Not estimated[h]
Utah, excluding Bear River region	None	None[i]	None	Not estimated[j]

[a] All estimates based upon meager data, both as regards topographic and geologic setting of individual deposits as well as chemical analyses. The estimates are of rock in the ground and no allowance is made for loss due to dilution or unfavorable structural conditions beyond those which can be judged from surface outcrops. All figures are rounded off to the nearest 25 million tons. Minimum mining width assumed to be 3 ft in both open-cut and underground mining.
[b] The only potential open-pit deposits known in Montana are in the Centennial Range, and in the area southwest of Lima. Though these deposits appear promising, not enough data are available to determine whether or not they can be mined by open-pit methods, or what their reserves may be.
[c] Estimated by M. R. Klepper; includes deposits in the Centennial Range, part of which may be amenable to open-pit mining.
[d] Estimated by M. R. Klepper.
[e] Minable rock containing more than 31 pct P_2O_5 is present in the Big Hole Mountains, where there may be a few million tons; rock containing 24 to 31 pct P_2O_5 may total several million tons.
[f] Reserves in both classes probably total scores of millions of tons.
[g] Modified from estimates made by R. H. King.[11]
[h] May total tens of millions of tons.
[i] A few small deposits near Provo are estimated to contain less than 1 million tons.
[j] May total a few million tons.

the amount of phosphate that might be available ultimately. For the most part only the high-quality rock was included in the estimates; in keeping with the quality of the data upon which the estimates were based, no refinements were introduced in the methods of calculation. These estimates served a very useful purpose in supplying legislators and mineral economists with data upon which to form policy on phosphate export, the administration of public lands, and other matters of national importance. But the early estimates are of little value in judging the potentialities of the field in terms of present mining practices and costs or in selecting deposits for mining.

Since mining activity has been expanded in the western field the Geological Survey has begun to acquire data which will permit estimation of reserves on a more realistic basis. Such estimates will not be completed for many years. Table 1, showing preliminary estimates of reserves of rock minable under present conditions, is presented with some reluctance, for—and it cannot be overemphasized—the estimates and data are incomplete.

The data available show that reserves of high-grade rock in deposits suitable for low-cost mining are limited indeed, known deposits contain a little more than 25 million tons, and it is improbable that new discoveries and appraisals will much more than double the figure. Reserves of high-grade rock suitable for underground mining (above entry level) are likewise surprisingly limited, 350 million tons in known and appraised deposits. Large tonnages, perhaps equaling or exceeding that of the Bear River region, of high-grade rock in beds 3 to 4 ft thick are present in the mountainous area of western Wyoming. But, making due allowance for rock in as yet unappraised deposits, it is highly unlikely that the western field as a whole contains more than a billion tons of rock containing more than 31 pct P_2O_5 in deposits minable by present practices. In fact, if the cost of transportation and underground mining persists at the present level in the future, it is probable that more favorably situated lower-grade deposits will be mined far in advance of the largest part of the potential billion tons of high-grade rock present over the field.

A much larger tonnage of rock containing 24 to 31 pct P_2O_5 is known. For example, 75 million tons in deposits suitable for stripping and a billion tons

suitable for underground mining are estimated in the Bear River region alone; this quantity may well be quadrupled when deposits in the remainder of the field are appraised.

The tonnage of rock containing more than 18 pct P_2O_5, though not yet estimated in even preliminary fashion, is vast indeed, and may be measurable in billions, perhaps even tens of billions of tons. For example, the southeast flank of the Uinta Mountains, near Vernal, contains several hundred million tons of rock with slightly more than 20 pct P_2O_5, most of which can be mined by open-cut methods. In the Bear River region the whole phosphatic shale member of the Phosphoria formation averages nearly 12 pct P_2O_5; and a thickness of about 70 ft, in the lower and upper zones together, averages 20 pct P_2O_5 (Fig 10). If these units were mined and beneficiated, nearly two-thirds of the phosphate in the formation would be recovered and a tonnage of perhaps 3 billion tons would be available.

Additional large reserves of both high- and low-grade rock would be available below entry level if shaft mining were to become practicable. In short, potential reserves of phosphate in the region are vast, and probable advances in mining and metallurgical technology may be expected to expand our concept of what constitutes minable reserves. Thus, at the present rate of production it would be only a few years until reserves in open-cut mines in the Bear River region were exhausted; but every operating company has research in progress which promises to enable them to mine lower-grade rock on their own properties before the deposits now being mined are exhausted.

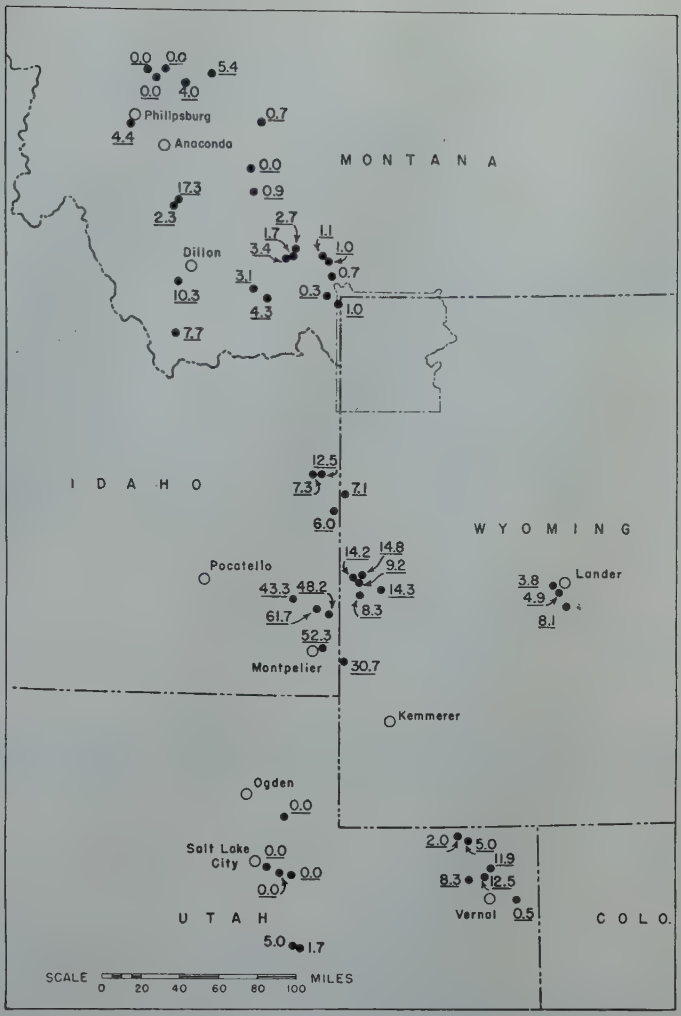


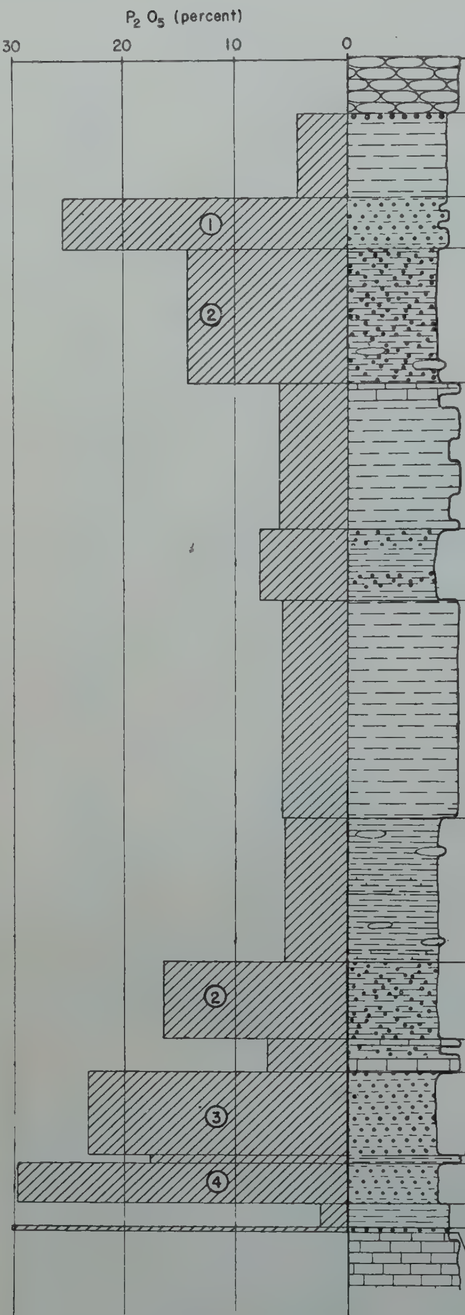
FIG 9—Total thickness (in feet) of rocks in the Phosphoria and Park City formations, containing more than 18 pct P_2O_5.

Conclusions

The western phosphate field may be divided into an eastern and western part. In the eastern part the geologic structure is simple, the folds are open and widely spaced, and the beds gently dipping; the phosphate deposits are for the most part thin and of relatively low quality. As a result of this structural pattern, outcrops of the Phosphoria formation are widely spaced, but are long and continuous; the cross-sectional area (along the dip of beds) above entry level is very large in individual areas and reserves of phosphatic rock are accordingly large, but of low quality; relatively large tonnages of rock amenable to open-pit mining are known and additional large tonnages will be discovered; underground mining over much of the area will be costly because of the thinness of the beds, although such increased costs may be somewhat offset by the lack of structural complexitites. The future of mining in the eastern part of the field lies in the discovery of areas where one or another of the phosphate beds is above average thickness and quality, as in the Centennial Range, and in the discovery and development of open-pit deposits where the cost of mining is low enough to offset the cost of beneficiating the lower grade rock.

In the western part of the field the geologic structure is complex; the folds are tight, closely spaced, and many of them overturned; the beds dip steeply, and faults of large and small displacement are abundant; the phosphate deposits, however, are thick and of high quality over much of the area, especially in the Bear River region. Outcrops of the Phosphoria formation are therefore closely spaced, but are short and discontinuous; though most individual deposits are small, total reserves are large because of repetition of the beds and the relatively great thickness of the phosphatic beds; only relatively small tonnages of high-quality rock minable by open-cut methods are available and few large additional deposits will be discovered. Underground mining of choice deposits may be expected to be as cheap as anywhere in the region, but will be expensive in many areas because of structural complexities. Additional valuable phosphate deposits may be discovered to the west in central Idaho, in the southwestern corner of Montana, in western Utah, and eastern Nevada; if the trend of westward thickening of the phosphate beds continues to those areas.

Our present understanding of the geology of the western field indicates that reserves of phosphate minable under existing economic conditions are



Thickness (feet)	P_2O_5 (percent) Av.	P_2O_5 (percent) Range	Thickness (ft.) X P_2O_5 (percent)	Lithology
				REX CHERT MEMBER
15	4.4	3.0 - 4.8	66	PHOSPHATIC SHALE MEMBER Mudstone: medium hard, brownish gray, thick-bedded; persistent nodular phosphatic rock, 0.5 ft. thick at top.
9	25.4	22.6 - 27.0	229	Upper phosphate zone: coarsely oolitic, medium-hard, brownish-gray, thin- and thick-bedded phosphate rock; thin layers of phosphatic, oolitic, soft, brown mudstone interbedded.
24	14.6	12.0 - 20.5	351	Mudstone: phosphatic, soft, brownish black, thin-bedded; increasingly vanadiferous toward top; calcareous lenses, up to 1 foot in thickness, abundant in lower part.
26	6.1	4.5 - 7.8	159	Upper nodular zone: medium-hard and hard, moderate brown, thick-bedded, mudstone; and interbedded thin layers of nodular, soft to hard, grayish-black phosphate rock. Persistent hard, massive limestone, 2 ft. thick, at top. Dark yellowish-orange and strong brown stains generally abundant.
13	8.0	6.6 - 8.6	104	Mudstone and phosphatic mudstone: soft, brownish-gray, thin-bedded mudstone; contains some phosphatic layers.
39	6.0	4.4 - 8.4	234	Middle nodular zone: silty, hard, moderate brown, massive mudstone and interbedded thin layers of nodular, hard, grayish-black phosphate rock. Dark yellowish-orange and strong brown stains generally abundant.
26	5.9	4.6 - 6.5	153	Mudstone: soft, brownish-gray, thin-bedded mudstone; hard, brownish-gray massive limestone layers in middle and at base; limestone lenses abundant throughout.
14	16.9	15.0 - 18.5	236	Mudstone: phosphatic, soft, brownish gray, thin-bedded; contains a few limestone lenses.
6	7.3	4.3 - 10.0	44	Limestone and phosphatic mudstone: hard, gray, massive limestone at top and base (basal layer known as "false cap"); phosphatic, soft thin-bedded mudstone in middle.
15	23.5	21.9 - 25.1	357	Phosphatic rock: argillaceous, soft, brownish gray, thin-bedded. A parting, 2 ft. in thickness, contains only about 16 percent P_2O_5; remainder averages about 25 percent P_2O_5.
1	17.8	8.7 - 26.2	18	"Cap" limestone: phosphatic, argillaceous, hard, thick-bedded. Generally soft, clayey, and more phosphatic at the surface.
7	29.7	24.8 - 32.8	208	Lower phosphate bed: soft, brownish black, thin-bedded. Generally contains more than 31 percent P_2O_5 in operating mines.
4	2.4	2.0 - 3.0	10	Mudstone: clayey, soft, brown, thin-bedded.
1	30.6	29.9 - 34.0	31	Contact phosphate bed: crystalline in part, hard, grayish black, thick-bedded.
				WELLS FORMATION

FIG 10—Generalized section of phosphatic shale member of Phosphoria formation in the Slug Creek Quadrangle, Idaho.

The phosphate content is based upon analyses of samples collected from a crosscut on the 300 level of the Conda mine; and trenches in Trail Canyon and south Dry Valley. Unit 1 contains 10.7 pct of total phosphate in the phosphatic shale member; unit 2 contains 26.6 pct of the total; unit 3, 16.2 pct; and unit 4, the only bed generally mined at present, 9.1 pct.

large, but by no means as large as earlier estimates would indicate. As a source of phosphate to be mined in the distant future from lower-grade beds and from shaft mines, however, the Phosphoria formation contains an even larger supply than indicated by earlier estimates.

Acknowledgments

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Mining of Phosphate Rock at Conda, Idaho

By T. C. RUSSELL,* Member AIME

The Conda phosphate mine, eight miles north of Soda Springs in Caribou County, Idaho, was opened up by the Anaconda Copper Mining Co. in 1920. Except for brief periods, during the 20's and early 30's, the mine has maintained a continuous production, and 2,100,000 short tons of phosphate rock have been shipped to the company's phosphate fertilizer manufacturing plant at Anaconda, Mont., or to independent processing plants on the Pacific Coast; a small tonnage of the rock has been shipped to the Hawaiian Islands.

The Phosphoria formation on the Conda property consists of two, commercial-grade phosphate beds separated by 147 ft of black to brownish colored phosphatic calcareous shales interbedded with softer mudstones, clays, and occasional narrow seams of oölitic phosphate. The main lower phosphate bed in the series averaging 7.1 ft in thickness, is called the "footwall bed," which lies conformably on the Wells limestone formation, the latter often containing irregular bands of chert. The upper or "hanging-wall bed" of phosphate rock averaging 7.2 ft in thickness, is overlaid by thick quartzite beds, locally called the "Rex chert" member in the upper portion of the Phosphoria formation. (Fig 1.)

There are two anticlines on the property with axis striking approximately parallel to the mountain range beneath which extensive development on phosphate rock has been conducted in the past, and is now in progress. The phosphate beds have been eroded to a large extent from the tops of these anticlines, leaving at least three well-pronounced outcrops of the phosphate beds exposed within the boundaries of the Anaconda Copper Mining Company's properties. At present, mining operations in the Conda No. 3 mine are confined to the two phosphate beds along the east flank of the easternmost anticline, where the dips on the beds vary from 50° to 60° east, thereby affording favorable conditions for the sublevel mining system now in use. Mining operations formerly were confined to the footwall or lower oölitic phosphate bed because of its higher phosphate and vanadium contents (32 pct P_2O_5 and 0.29 pct V_2O_5); this bed averages 7.1 ft in thickness. The oölitic portions of the upper bed were of satisfactory grade, but the shale and clay partings (Fig 2) diluted the mined rock and could not be separated economically by selective mining or by sorting. Experiments in beneficiating this rock were carried on for a number of years. Recently, a simple washing process was adopted, which gave a satisfactory grade of concentrates and a high recovery of P_2O_5.

In 1947 it was decided to expand Anaconda's production of treble superphosphate and phosphoric acid by approximately 90 pct. This rock will be obtained from the footwall and hanging-wall beds, and the rock from the hanging-wall bed will be beneficiated at Anaconda.[1]

In order to take advantage of the previous development on the 500 secondary haulage level, short crosscuts will be driven at approximately 950 ft intervals from the existing lateral drifts through the hanging-wall shale beds. From the inner end of each crosscut, short lateral drifts will be driven in both directions, so that four working raises can be put up on the dip of the phosphate bed at proper intervals along each of the lateral drifts. The mining operation on the footwall bed at Conda

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[1] References are at the end of this paper.

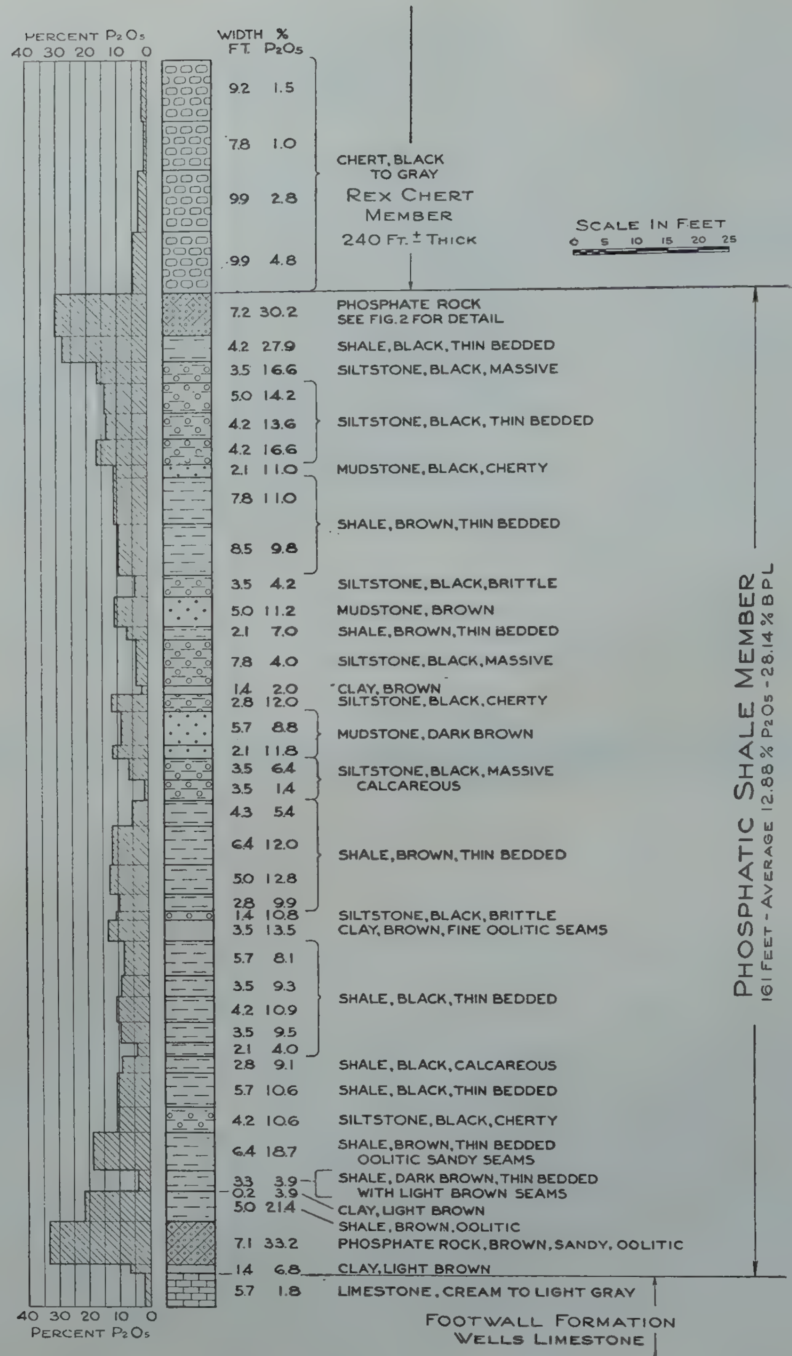

FIG 1—Columnar section of Phosphoria formation as exposed in 509 crosscut, No. 3 mine, Conda, Idaho, showing variable P_2O_5 content.

has been previously described.[2] This bed lies on the Wells limestone, which provides a firm footwall, but the phosphate rock is overlaid by thin bedded phosphatic shales, which disintegrate readily. The mining conditions in the hanging-wall bed are reversed; here the footwall is composed of soft shale and clay and the hanging-wall is formed by the hard massive Rex chert. These new conditions were satisfactorily met by using a sublevel mining system. Three compartment raises are driven on the phosphate bed spaced at 196 ft centers. The sublevels are driven from the raises at 30 ft intervals along the dip and extend for 90 ft in each direction, as drifts on the bed. Stoping operations are started by raising through the 21 ft pillar of ore from the farther ends of the sublevels. The rock is then stoped out in a succession of steps (Fig 3). The overhead gob is supported by the stull and plank flooring of the previous slice. The soft footwall is held in place by 2 by 10 in. pine lagging placed as footboards under the stulls, which are used for staging and for temporary support for the unmined portion of the slice.

The phosphate rock is drilled with pneumatic feed-leg jackhammers, equipped with downstroke rotation. Auger drill steel, 1¾ in. in diameter, is used. Bits are V-shaped and the cutting edges are faced with stellite. Holes are drilled 6 ft deep across the width of the bed and the broken ore is blasted down to the sublevel and

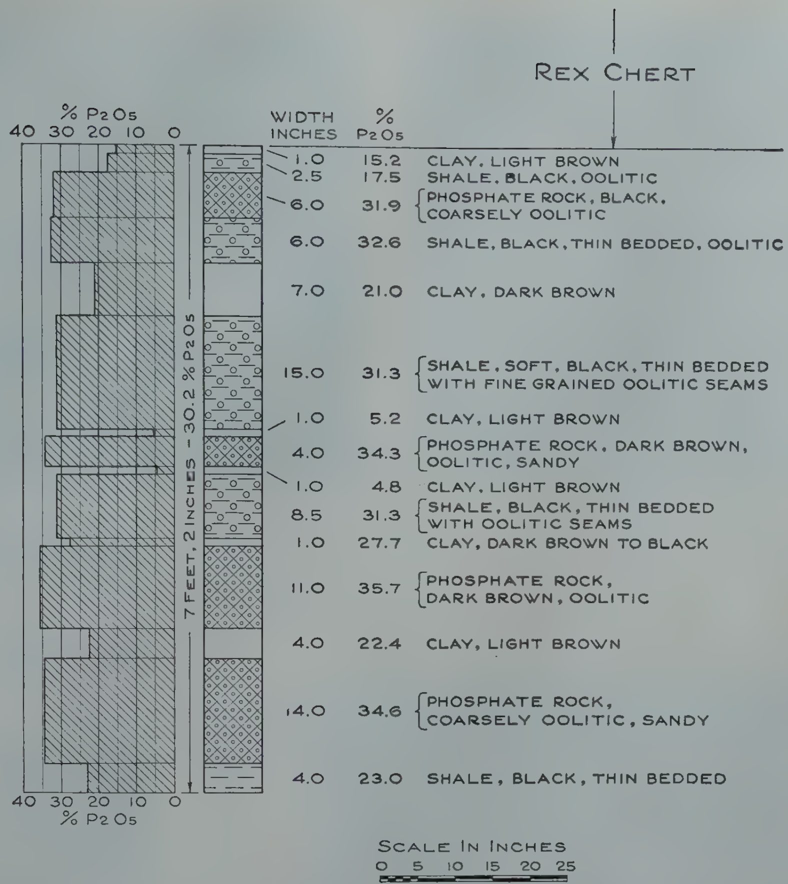

FIG 2—Columnar section, detail of hanging-wall bed of the Phosphoria formation, as exposed in 509 crosscut of No. 3 mine, Conda, Idaho, showing P_2O_5 content.

slushed into the ore chutes on either side of the working raises. Anaconda-type scrapers of 10.5 cu ft capacity are used for slushing. A 15 hp, two speed, electric hoist is set up in the manway compartment of the raise to pull the scrapers from both directions. The slushing hoists are also used for hoisting timbers and other material. The rock is loaded into 2½-ton capacity side dump cars. The cars are hauled by 4-ton storage battery locomotives from the raise chutes to the station ore pockets in strings of 15 cars. All of the mine haulage tracks have a standard gauge of 3 ft. At the shaft station three cars can be dumped at each spotting of the train over the station ore pocket, from which the ore drops through an inclined pass into one of the vertical rock transfer compartments of the shaft (Fig 4). About 500 ft below the station the rock flows into one of the three cylindrical loading bins, which have been excavated above the main haulage level. Each of the bins has a capacity of 400 tons and is provided with four air-lift

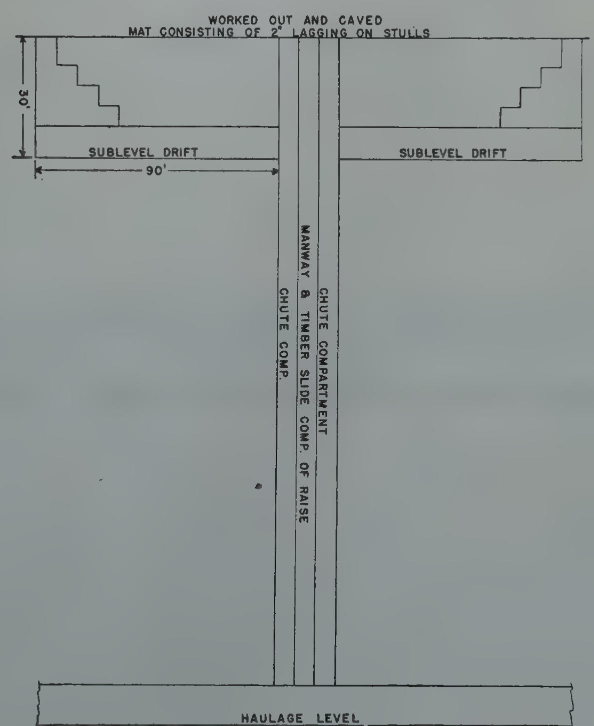

FIG 3—Long section of raise and sublevel stopes on hanging-wall bed at Conda, Idaho.

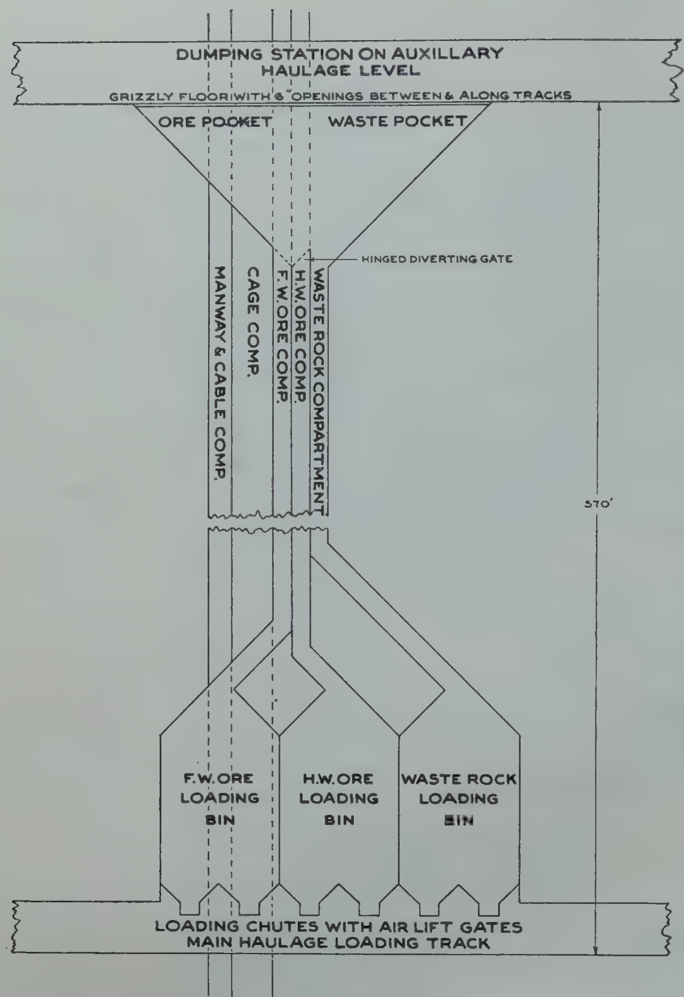

FIG 4—Elevation of shaft at Conda, showing arrangement for handling two classes of phosphate rock, and waste from auxiliary haulage to main haulage level.

gates, which facilitate quick loading. The shaft compartments, stations, ore pockets, passes, and loading bins are lined with from 8 to 10 in. of reinforced concrete. From the loading bins the hanging-wall rock, footwall rock, and waste is loaded into side dump cars of 11 tons capacity. These cars are trammed in ten car trips, by 20-ton storage battery locomotives, a distance of 8500 ft to the surface mill bins or to the waste dump.

At the mill the mine-run phosphate rock is screened over ⅜ in. openings and the oversize is crushed in closed circuit through a swinging hammer pulverizer.

During the winter months the rock is dried from 6 to minus 2 pct moisture to prevent freezing in the railroad cars, while in transit to the phosphate plant at Anaconda. The present daily production is approximately 590 wet tons on a six day per week operation.

Acknowledgments

The author wishes to thank Mr. Edward Shea of the Anaconda Copper Mining Co., Geological Department at Butte, and Mr. Martin J. Ruggles, Mining Engineer, Anaconda Copper Mining Co., Fertilizer Department, Conda, Idaho, for their assistance in preparing sections and drawings.

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Anaconda Phosphate Plant, Beneficiation and Treatment of Low Grade Idaho Phosphate Rock

By R. J. CARO*

The Anaconda phosphate plant was put into operation in the fall of 1923. Its present daily operating capacity is approximately 170 tons of treble superphosphate and 16 tons of phosphoric acid analyzing 42 pct P_2O_5 and 52 pct P_2O_5, respectively. As a result of the continued increased demand for phosphates, Anaconda decided in 1947 to expand the plant's daily output to 270 tons of treble superphosphate and 54 tons of phosphoric acid. The present daily plant intake is 335 tons of wet (footwall) phosphate rock analyzing 31.7 pct P_2O_5 and 0.28 pct V_2O_5. The rock is shipped from Anaconda's mine at Conda, Idaho, in open railroad cars, having been crushed to pass ⅜ in. mesh at the mine.[1] The expansion program will require an increased intake of 300 tons of rock per day. The hanging-wall rock analyses approximately 30.5 pct P_2O_5 and 0.11 pct V_2O_5, and will be beneficiated in a new washing plant under construction

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* Superintendent, Phosphate Plant, Anaconda Copper Mining Co., Anaconda, Mont.
[1] References are at the end of this paper.

FIG 1—Lurgi filter belt.

FIG 2—Treble-superphosphate setting belt.

in the main phosphate plant building at Anaconda.

Washing Plant Flowsheet
Hanging-wall phosphate rock, minus ⅜ in.
20 to 30 tph
↓
Scouring mill, 10 ft by 48 in.
↓ Wash water
Dorr classifier, 4 ft
↓ Slime — Sand ↓
Waste ← Slime; Natural gas
Sand → Rotary drier, 5 by 20 ft
↓
Elevator; Natural gas
↓
2 wedge type roaster furnaces, 20 ft
↓
Raymond roller mills

FIG 3—Anaconda phosphate plant at Anaconda, Mont.
Photograph by courtesy of Mining World.

The function of the scouring mill is to break up clay masses and to free the fine-grained shale particles having a high alumina and iron but low P_2O_5 content. This mill will operate at 15 rpm and will be equipped with interior lifts. In the washing operation, approximately 1.17 tons of hanging-wall rock of 30.7 pct P_2O_5 grade will produce 1.0 ton of sand of 33.7 pct P_2O_5 grade, giving a P_2O_5 recovery of 93.8 pct.

Table 1 . . . Typical Results Obtained Washing Conda Hanging-wall Rock

Per Cent	Crushed Rock	Sand	Slimes
P_2O_5	30.50	33.70	13.10
Al_2O_3	2.09	0.92	8.50
Fe_2O_3	1.37	0.74	4.60
SiO_2	13.00	7.90	39.80
H_2O	1.40	16.00	94.00

The scouring mill discharges at 70 to 80 pct solids. Control is based on the maintenance of a slime pulp density at about 1.04 to 1.06. Fresh water is put on deck of classifier to promote washing of the slimes from the sand. Due to packing of sand in bins, moisture content of sand is reduced from 16 to 6 pct in rotary drier ahead of calcining furnaces.

The crushed, undried "footwall rock," as received from Conda, is first calcined at a maximum temperature of 1675°F in four wedge-type, gas-fired roaster furnaces in order to burn off the excess organic material (9.0 pct). The beneficiated rock will receive the same treatment. Two additional 20 ft wedge-type furnaces will be put in operation for this purpose. The calcined rock will be ground to pass 93 pct through 100 mesh in three No. 6669 low side Raymond roller mills. Each of these mills has a capacity of approximately 12 tons of rock per hour.

The high vanadium (footwall rock) product from the roller mills goes to two Pachuca-type agitators, 15 ft in diameter and 30 ft high, operated in series. In the agitators the rock is mixed with 60° Be sulphuric acid together with weak phosphoric acid wash solution from the Lurgi filters (Fig 1). Phosphoric acid and gypsum are produced. The phosphoric acid contains vanadium compounds in solution. Phosphoric acid slurry from the agitators is passed over three 5 meter improved type Lurgi filters, where gypsum cake is washed and sluiced to waste. The Lurgi filtrate, having 27° Be strength, goes to five single effect Swenson evaporators.

Two products are obtained from the evaporators, one contains approxi-

mately 46 pct P_2O_5 and the second 52 pct P_2O_5. The acid products are pumped to six 250-ton capacity cone-bottom tanks for removal of fluosilicate salts and clarification of the acids. Clarified acid is sent to the vanadium recovery plant for the removal of this element in the form of vanadium pentoxide.[2]

The devanatized 52 pct P_2O_5 acid is shipped to consumers in rubber-lined steel tank cars, and is used as fertilizer through application to the soil by titration into the irrigation water. The devanatized 46 pct P_2O_5 acid is mixed with the low vanadium (hanging-wall rock) product from the roller mills in a model 104, 2-ton Pratt mixer. Discharge from the mixer drops onto a 48 in. rubber setting belt, 529 ft in length and traveling at a speed of 8 fpm (Fig 2), the wet material remains on the belt for 29 min during which time it takes an initial set. The setting belt discharges into a 55 by 167 ft curing shed, equipped with a traveling crane and clamshell bucket. Curing continues for 20 days, during which time the chemical action raises the available P_2O_5 content of the treble superphosphate and improves its physical characteristics.

The cured treble superphosphate from the shed is then dried in four wedge-type gas-fired roaster furnaces, after which it is crushed in 2 H. R. Sturtevant ring-roll grinders in closed circuit with Tyler Hummer-type vibrating screens equipped with screen cloth having an opening of 1/8 in. The undersize goes through a 6 by 10 ft rotating drum equipped with lifts for dedusting. A rapid stream of air is drawn through the cascading material and essentially all of the minus 100 mesh product is removed and caught in 110 wool bags which are 18 by 30 ft. These fines are then pelletized by moistening with water in a 6 by 28 ft revolving drum and dried in a 6 by 40 ft gas-fired brick-lined drier, after which they pass through a 3.5 by 15 ft cooler and are sized over a Jeffery screen, the fines go to finished product and the oversize is returned to the ring-roll grinders.

The final product is shipped to consumers in both bulk form and in 100 lb 5-ply paper bags. A screw feed device designed at Anaconda is used for the rapid loading of cars in bulk shipments. A model "A" Bagpacker fills, sews, seals, and conveys 15 bags per minute when bagged shipments are made.

In order to augment the present plant production of 235 tons of 60°Be sulphuric acid per day, a 150 ton Chemical Construction Co. vanadium catalyst contact acid plant has been constructed adjacent to the phosphate plant roasters. Iron sulphide floatation concentrates, containing about 42 pct sulphur, will be obtained from the copper concentrator tailings, and calcined in five wedge-type roasters to produce the SO_2 gas. Calcines will be sluiced to waste.

With only minor changes all additional equipment units required to meet the demands of the expanded program will be duplicates of equipment now in operation. These additions include two Pachuca-type agitators, two Lurgi filters, five Swenson evaporators, eight 250-ton capacity wood-stave, lead-brick lined acid clarification tanks. A 2-ton Pratt mixer, one 48 in. setting belt 529 ft long, curing shed with traveling crane, two 20 ft wedge-type drying furnaces for treble-super-phosphate, a No. 2 Sturtevant ring-roll grinder, deduster, and a Sly dust control installation for entrapment of fines from the second deduster. Fig 3 shows the phosphate plant at Anaconda.

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Surface Strip Phosphate Mining at Leefe, Wyoming, and Montpelier, Idaho

By D. L. KING*

The San Francisco Chemical Co. has been actively interested in phosphate mining since 1908. It was, in fact, the first company to make claims on properties in the western phosphate belt. From the period of 1908 to 1947, the company's phosphate operations were centered at their Waterloo property, five miles east of Montpelier, Idaho. A small tonnage of phosphate rock was shipped annually from their underground mines in that area. In 1945, the company started the first western open-pit phosphate mine on the slopes of Waterloo Hill. In 1947, they obtained an option and lease on the Beckwith Hills' deposits in southwestern Wyoming, some 25 miles south of Cokeville. While the operations at Montpelier and the Beckwith Hills are both open pit, there are many differences which suggest separate treatment in this paper.

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* San Francisco Chemical Co., Montpelier, Idaho.

Waterloo Hill Operations

Structurally, the Waterloo Hill is the west slope of an anticline. The beddings dip from 20° to 35° to the west. Erosion has caused the footwall exposures (Well's formation) to appear in the upper areas as a series of apices, giving the outcrops a festooned arrangement. Dip faulting with subsequent erosion has induced gullies normal to the strike and resulted in eroded areas which separate the remnants of the

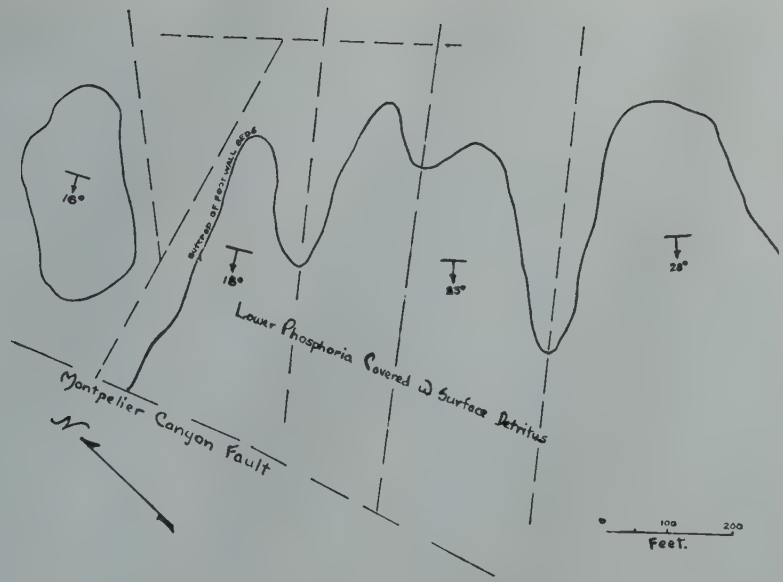

FIG 1—Plan view of Waterloo Hill showing outcrop pattern.

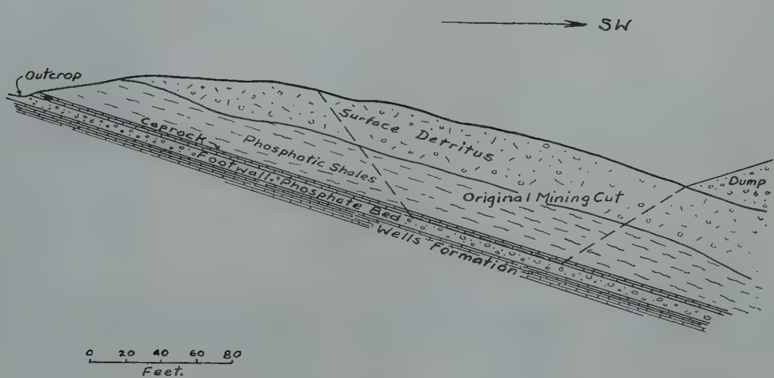

FIG 2—Sectional view of Waterloo Hill.

Phosphoria (see Fig 1). The Rex chert and several of the upper phosphatic beds are completely absent on the Waterloo Hill slope so that the overburden consists of the lower phosphatic beds together with some alluvium. Because the beds dip at a slightly steeper angle than the slope of the hill there is increased overburden as advance is made down the slope (Fig 2).

The entire hill slope area was diamond drilled first to determine the remnant and the completely eroded areas; then, once these were defined, coordinate drilling was employed with holes at 250 ft staggered pattern. Thus starting benches were determined as well as lateral spoil areas. Core recovery was very poor, but determination of horizon by drilling was clear cut due to the existence of the limestone cap rock which, fortunately, is very fossiliferous at its base and thus is an excellent marker element.

The point along the slope at which overburden could be economically removed from the footwall bedding was determined and a bench started at that point. A stripping area large enough to allow for the recovery of 20,000 tons of phosphate was stripped from the first panel. All stripping was accomplished by means of 18 to 20 yard scrapers pulled by D-8 caterpillar tractors. The first stage of stripping was carried from the grass roots to the cap rock immediately covering the footwall phosphate beds. Some ripping was necessary in order to effect the stripping of some limestone beds occurring within the phosphatic series. On low slopes "push cats" were employed to load the scrapers. As soon as an initial bench was established and a sizable area of cap rock exposed, power shovels were started along the upper edge of the exposed area to remove the limestone cap rock of fine-grained limestone, approximately 22 in. thick and sufficiently tight to preclude economical removal by any other method. The cap rock removed from the upper shovel cuts was overcast and dozed away to the spoil pile by bulldozers. As soon as sufficient area was decapped, a small power shovel was placed in operation, loading the exposed phosphate into trucks for

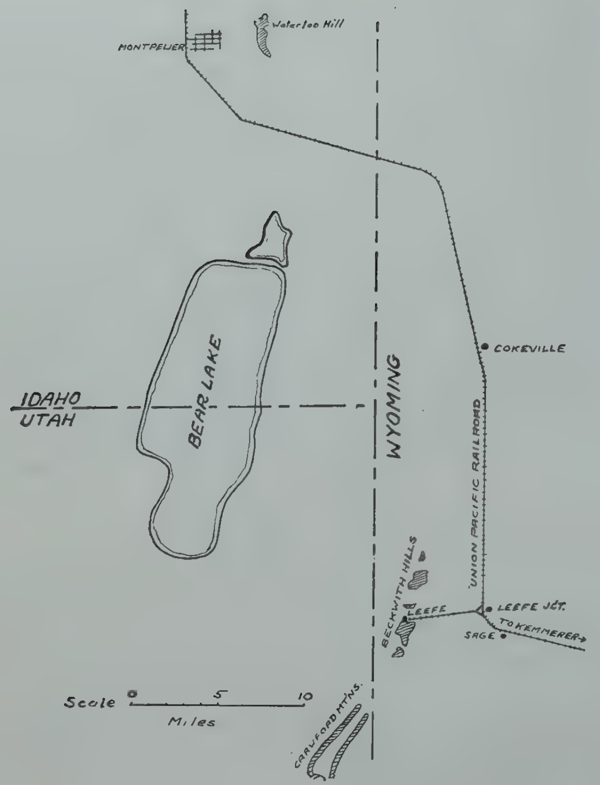

FIG 3—Waterloo Hill, Beckwith Hills, and Crawford Mountains phosphate deposits.

haulage to the rail head at Montpelier. Decapping continued simultaneously with the removal of the phosphate until the entire exposed area had been decapped. Overburden removal progressed simultaneously with these operations at the same level and adjacent to the initial bench. As soon as the initial panel had been decapped and the phosphate removed, stripping was begun in the area immediately above the panel. This procedure, which makes possible the wasting of subsequent panel overburden into the mined out area, was followed until the upper limit or the east outcrop was reached.

In twenty months of operation, beginning in 1945 and continuing until 1947, 650,000 tons of commercial grade phosphate (31.5 pct P_2O_5) was extracted from the Waterloo Hill by the above mining practices.

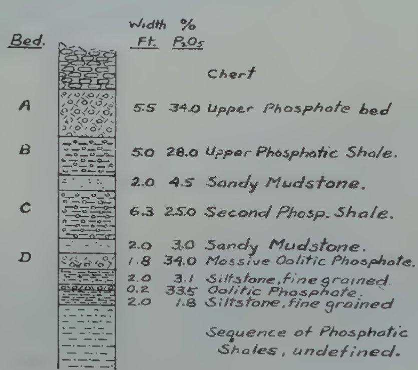

FIG 4—Generalized section of upper phosphatic shale sequence at Leefe, Wyo.

Wyoming Operations

The Wyoming operations of the company are centered in the Beckwith Hills area about 3½ miles west of Sage, Wyo., (a station on the Union Pacific main line about 25 miles south of Cokeville, Wyo., and 25 miles west of Kemmerer, Wyo.). A railroad spur was constructed to connect the mill and camp site with the Union Pacific main line about one mile west of the Sage, Wyo., station. This junction, as well as the mill and camp site, is known as Leefe, Wyo. (Fig 3).

The Beckwith Hills are a low lying end-phase of the Crawford Mountains, which appear in bold relief some three miles to the south. The Hills are part of the western limb of a gentle syncline, with an approximately north-south axis. The structure has a slight dip to the west. A sharp strike fault occurs along the eastern edge of the Hills, and along the western edge the partly eroded remnants of a tight overturned fold can be found. The Phosphoria formation outcrops along the eastern and western edges of the hills with a north-south strike. A system of east-west faults has induced erosion gullies across the strike of the beds and cut the residual Phosphoria into a series of elliptically shaped sections, the major axes of which are east-west (Fig 4). So far as is now known, these sections are separated by areas of footwall limestone and quartzite (Weber formation, the local correlation of the Wells). Because of this erosional effect, the mining panels are patterned to the residual sections, around which the outcrops of the top bed of the Phosphoria can be easily traced. Present mining practice is to strip the overburden around the fringes of each section beginning at the outcrop and extending inward for a distance determined by the depth of overburden. This practice is designed to reduce the center of mass haul to the minimum, all spoiling being confined to the areas immediately outside and below the natural outcrop line. Due to the gentle dipping of the beds, this practice produces a symmetrical stripping and mining picture when viewed aerially.

In contrast to the other western phosphate operations, the uppermost phosphatic bed is the first objective (locally, this bed is spoken of as "hanging-wall bed"), all other active western operations have as their primary objective the "footwall bed." The phosphate rock occurring in this upper bed is distinctly different from all other western productive beds in that it is a light colored, massive high-grade rock containing from 33.5 to 34.7 pct P_2O_5. Its light color, contrasted to the other western phosphate rocks, is apparently due to a long period of leaching and oxidation which has resulted in the impoverishment of the carbonaceous content which is characteristic of western phosphates. Its massive characteristics are common in this specific area (as in the Crawford Mountains to the south) due, no doubt, to severe post-deposition compression. The characteristic oölitic grain texture is apparent upon close examination, but the severe compression has resulted in nearly obscuring the individual oölites to such a degree that in polished

section only can the oölitic texture be recognized.

Directly below the hanging-wall or *A* bed lies a sequence of four phosphatic shale beds designated and described respectively as: *B* bed, a phosphatic shale member 60 in. in thickness and averaging 28.0 pct P_2O_5 (there is no parting material between *A* and *B* beds); *C* bed, a phosphatic shale member 76 in. in thickness and averaging 25 pct P_2O_5 (*C* is separated from *B* by a sandy mudstone layer 24 in. in thickness and averaging 4.5 pct P_2O_5); *D* bed, a massive oölitic phosphate rock 22 in. in thickness similar in physical appearance and phosphatic content to *A* bed (*D* bed is separated from *C* bed by a low-grade sandy mudstone 24 in. in thickness and averaging 3.0 pct P_2O_5).

Mining and recovery of the hanging-wall bed will proceed as outlined until all of the fringe areas of the various mining panels are thus stripped and mined, by which time it is expected a treatment plant will have been built and placed in operation for the upgrading of *B* and *C* beds. At that time these latter beds will be mined and beneficiated to produce a rock suitable for the fertilizer trade. The selective stripping and recovery of *D* bed has not been decided upon yet, but it is probable that it will be mined and treated in conjunction with *C*. As soon as *B*, *C*, and *D* beds have been removed, the former fringe areas will then be available for wasting overburden from the second cut. The identical sequence will be practiced on the subsequent cuts until the mining panels have become exhausted.

Definition and development of the mining panels has been accomplished by a program of diamond drilling which involves spacing of vertical holes at 200 ft centers. Because of the loose chert fragments, diamond drilling of the surface mantle was extremely difficult, and core recovery of the phosphatic beddings was very poor. The only practical value of the drilling has been to define the horizon of the hanging-wall bed with relation to the surface. Drilling to obtain samples was entirely unsuccessful.

The surface mining plant consists of: 1 Bucyrus-Erie 54-B shovel, 1 Link-Belt K-570 shovel, 1 Link-Belt K-370 shovel, 1 Bucyrus-Erie 37-B shovel, 1 Bucyrus-Erie 1030 shovel, 2 Unit 1020 shovels, 4 27FD Euclid trucks, 3 D-8 Caterpillar dozers, 2 Le Tourneau 12 cu yd buggies, 4 KBS-7 International dump trucks, 4 Chevrolet 2-ton dump trucks, 1 315CFM portable compressor, 2 wagon drills, and other complementary equipment. Other plant facilities consist of a primary crushing unit, a secondary crushing and screening unit, stock piling and loading facilities for all sizes of crushed and graded rock, crushed rock storage, and Raymond mill drying and pulverizing equipment. The plant has a capacity of approximately 150 tph of minus 1 in. crushed rock and of 25 tph of rock dried to 1 pct moisture and pulverized to 93 pct minus 100 mesh. Direct rail service is effected by the company spur, and complete camp facilities are near the shipping site.

Mining Operations of the Montana Phosphate Products Company

By R. J. ARMSTRONG,* Member AIME, and J. J. McKAY*

The Montana Phosphate Products Co. is currently operating three underground phosphate mines in Powell County, Mont.†

In this area the Phosphoria formation is from 35 to 50 ft thick and contains a bed of phosphate rock varying in thickness from 2½ to 5 ft, overlain by 12 to 30 ft of cherty quartzite, and underlain by about the same thickness of quartzite and shaly to cherty beds. In general, the phosphate bed has a reasonably hard quartzite footwall, as well as a hard cherty hanging wall, both of which greatly facilitate the mining operation.

Folding has deformed the phosphate bed to a considerable degree in the area. It is carried to surface by several of the anticlines and lowered to considerable depths in some of the synclines. The main structural features affecting the mining operations of the company are the Garrison anticline and the adjoining syncline and succeeding limb of a second anticline on the east (Fig 1). The axis of the folding strikes roughly northwest-southeast.

The three mines, namely, the Anderson, Graveley, and Luke are on different limbs of the above folds. Accordingly, the dip of the bed, and consequently the mining method, in each mine differs to a considerable degree from that of the others. For this reason, each operation will be dealt with separately.

Anderson Mine

LOCATION AND DEVELOPMENT

Topographically this mine is in a narrow valley, the bottom of which has an altitude of 5200 ft. The phosphate bed outcrops near the ridge on the east and dips towards the valley floor at an angle of from 25 to 30°,

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* Superintendent and Assistant Superintendent, respectively, Montana Phosphate Products Co., Garrison, Montana.
† See also paper by Geoffrey Gilbert: Mining Operations of the Montana Phosphate Products Company. TP 1824, *Min. Tech.* (May 1945); *Trans.* AIME (1947) 173, 240.

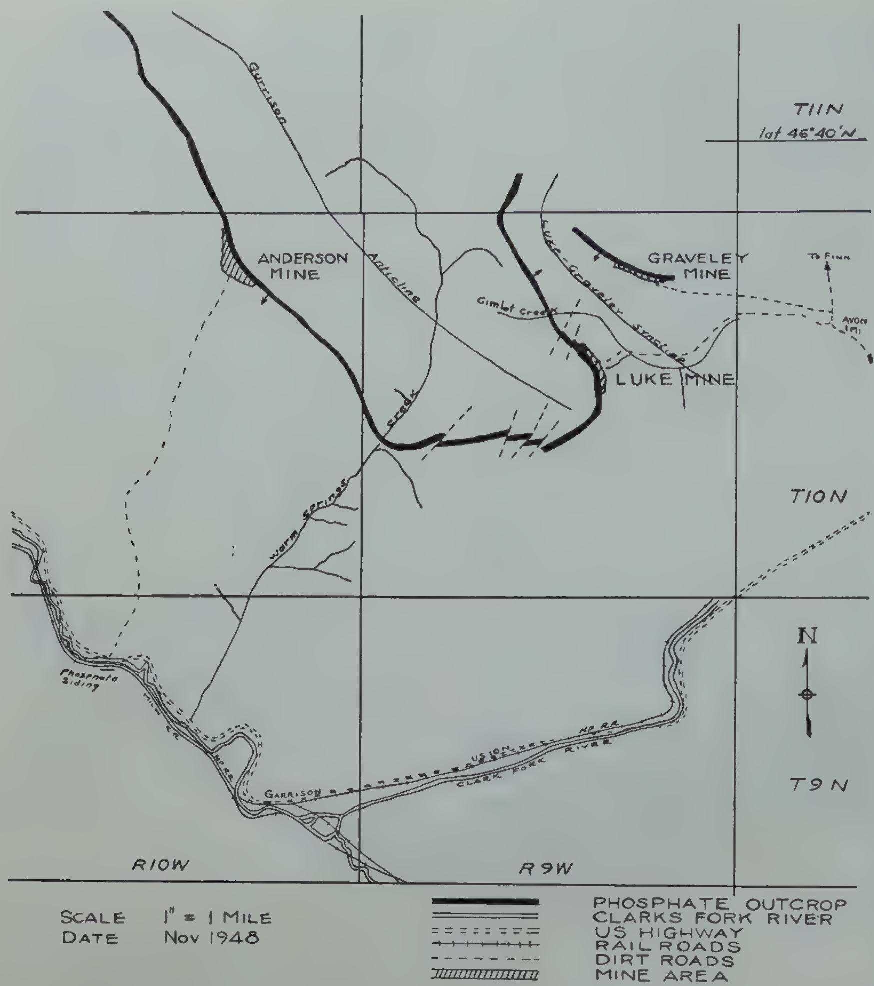

FIG 1—Location of phosphate operations, Montana Phosphate Products Co.
(Reduced approximately 50 pct in reproduction.)

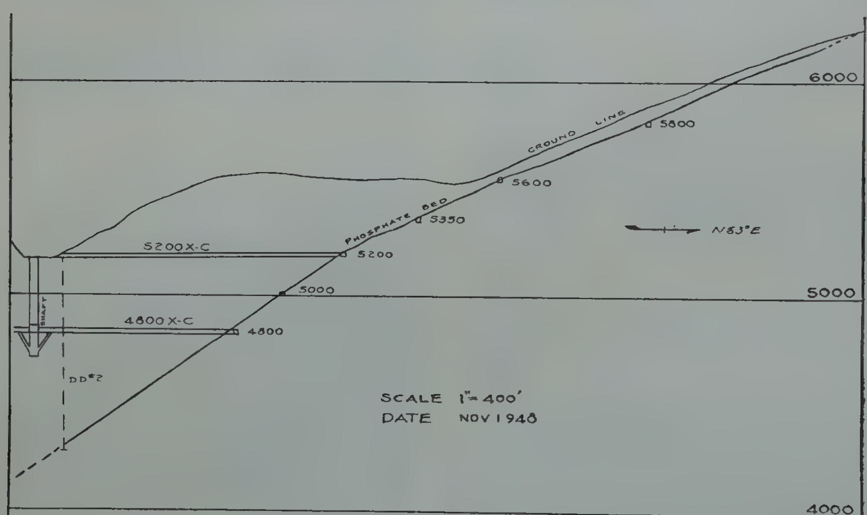

FIG 2—Cross section of phosphate bed at Anderson mine.
(Reduced approximately 40 pct in reproduction.)

which is slightly steeper than the inclination of the hillside. The strike is more or less parallel to the direction of the valley.

The mine was developed originally by tunnels driven in the hillside through the hanging-wall formation at regular level intervals. All ore above the floor of the valley was developed in this manner, but to develop the downward extension of the bed below the valley floor, it has become necessary recently to have a shaft (Fig 2). The shaft was sunk vertically through the hanging-wall formation to a depth of 475 ft, which is sufficient for the development of two levels with necessary room at the bottom for loading pockets and sumps. It is 7 by 17 ft outside the timbers and contains three 4½ by 5 ft compartments, two for hoisting and one for pipe and manway. Drifts and crosscuts are 8 by 8 ft in cross section. Drifting is done in the quartzite footwall immediately below the phosphate bed. Drift rounds are drilled with 3½ in. drifters mounted on Sullivan single jib drill carriages and mucking is done with Eimco 21 mucking machines.

STOPING

Chute cutouts are made at 72 ft centers along the drift and are driven in the phosphate bed a distance of 20 ft up the dip before the chute is installed. The chutes are of timber construction, 3 ft wide at the lip and fanned out to accommodate a 5 ft scraper at the upper end. Wingboards are installed from the upper end of the chute to the pillar on either side. Because of the flat dip of the bed, manways are not necessary and access to the stope is gained over these wingboards.

An open stope, or room-and-pillar method of mining, is used; the stopes are carried 60 ft wide and are separated by a 12 ft pillar from the adjoining stope on each side (Fig 3). These pillars are broken through at 20 to 25 ft intervals to provide better ventilation and secondary exits. Although for the most part the hanging wall is sound and stands up well, as a safety precaution two stull rows 20 ft apart with stulls at 10 ft centers are carried up the center of the stope.

The dip of the bed at 25° to 30° is too flat for the broken ore to run by gravity from the stopes to the chutes and scraping is necessary. Double drum slusher hoists of 15 hp are installed at the chutes on platforms above the drift. These operate V-type scrapers which are 5 ft long, 18 in. high, and have a capacity of about 10 cu ft. On the hoists ⅜ in. haulback and 7/16 in. load cables are used. The cables run through an 8 in. diam haulback block hung on an adjustable steel stull which can be moved readily across the face of the stope in order

that the scraper can clean out the ore across the whole width.

A stope crew consists of two miners on one shift and a scraper man on the opposite shift. As the phosphate rock is quite soft, the miners generally drill and blast the entire 60 ft face in one shift. This requires from 40 to 50 6-ft holes and on an average advances the stope face about 5 ft. Drilling is done with light 2¾ in. stopers using 1 in. quarter octagon steel and detachable bits. A set of bits will generally drill about 10 holes before requiring reconditioning. The broken rock is scraped to the chute and into the ore train by the scraper man on the opposite shift. It has been found that the economical limit for scraping at this dip, and under the described conditions, is about 300 ft. The level interval is, therefore, laid out to give this slope distance on the bed.

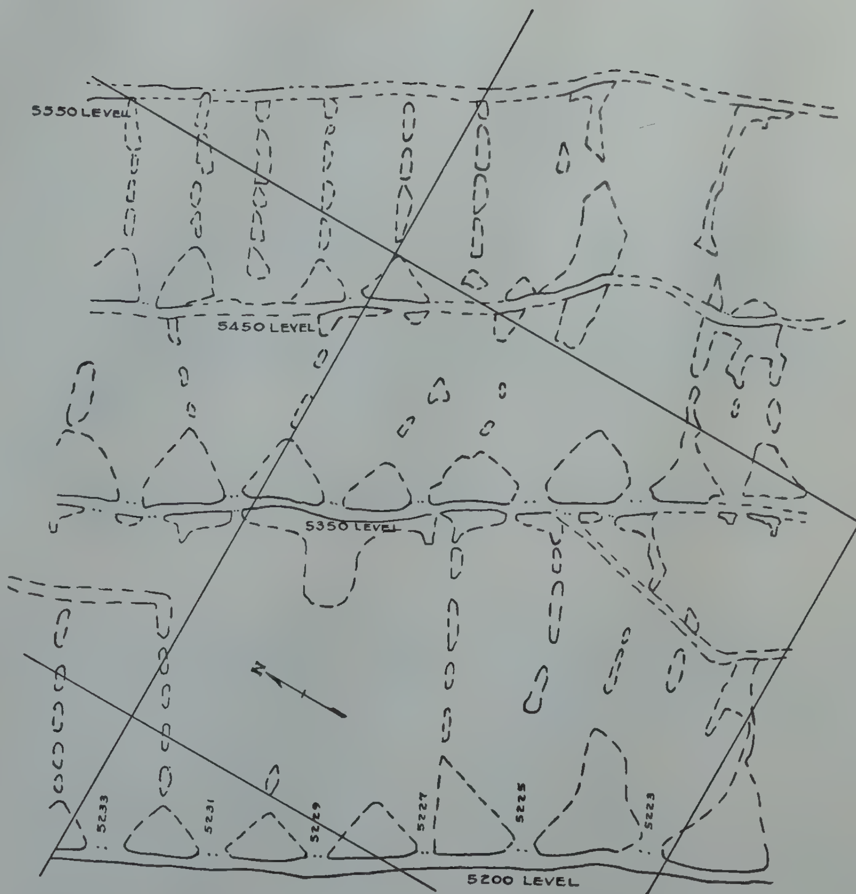

FIG 3—Plan showing room and pillar method of stoping at Anderson mine.

UNDERGROUND TRANSPORTATION AND HOISTING

Broken ore is hauled from the stopes to the shaft pockets, or in the case of levels developed by tunnels directly to the surface bunkers, in 33 cu ft Coeur d'Alene type side dump cars. Ten cars make up a train which is powered by a Mancha Titan AX, 3 ton, battery locomotive. Ore is hoisted from the shaft pockets in two 50 cu ft, side dump skips operated in balance. The skips discharge into a 400 ton surface bunker adjoining the headframe (Fig 4). From the surface bunkers the rock is transported by truck seven miles down a well-maintained, gravel road to railroad cars for shipment to the fertilizer plant.

Luke Mine

LOCATION AND DEVELOPMENT

The Luke mine is about 6 miles southeast of the Anderson on the opposite side of the Garrison anticline. The terrain between the two mines is rugged and road communication from the Anderson is through Garrison to Avon along highway 10 and thence along the county and finally the mine road to the Luke mine, a total distance of 32 miles. A forestry-type phone line connects the Luke and Graveley with each other and with the main office at the Anderson. The Luke camp is in a thickly wooded draw which slopes quite steeply down to Gimlet Creek (Fig 1).

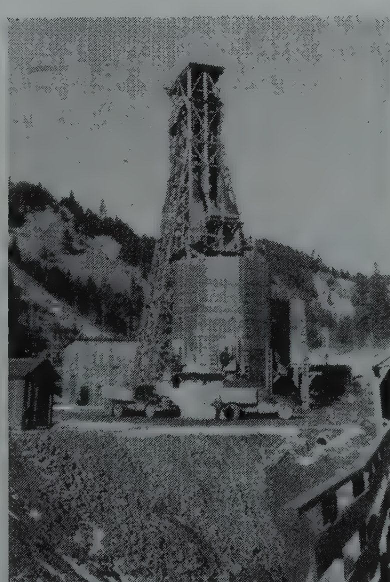

FIG 4—Anderson mine head frame and ore bin.

The Luke mine is somewhat younger than the Anderson, over half of the tonnage shipped being produced during the last two years. Development was commenced by driving a drift along the bed at 5700 ft elevation from an outcrop near the portal. All production from above this level has come through the 5700 level portal, although two short lengths of intermediate drifts have been driven at 5900 and 6050 ft elevations to service the upper workings.

In October 1946, an 8 by 8 ft crosscut adit was started near Gimlet Creek at 5300 ft elevation. This adit was driven 2700 ft and intersected the bed in July 1947. The portal is in Kootenai formation while the last 600 ft before encountering the Phosphoria formation was driven through the Ellis shales and sandstones. From a mining standpoint it is interesting to note that for the most part the Kootenai proved soft and had to be timbered while no timber was required in the Ellis. Small flows of water were encountered in nearly all the numerous sandstone beds cut by the adit, but no water was found in the Phosphoria.

The Phosphoria at the Luke consists of a phosphate bed about 3 ft wide, overlain by about 26 ft of quartz-

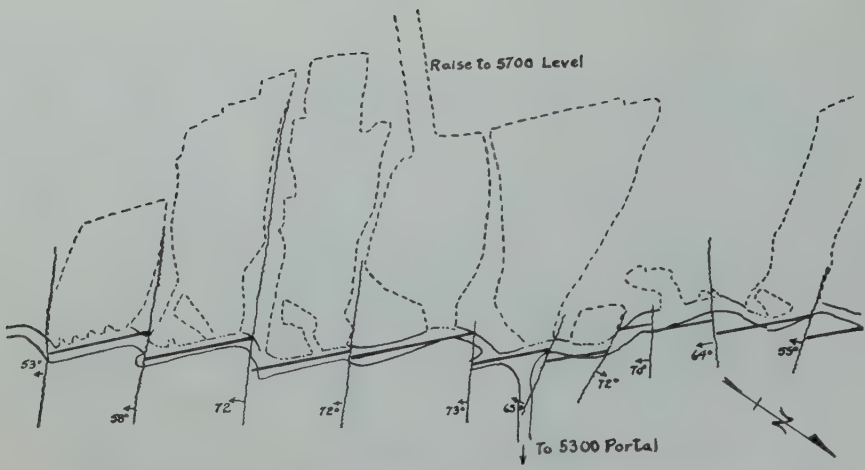

FIG 5—Plan showing effect of faulting on drifting and stoping at Luke mine.

ite or chert. The dip is about 45° and, as at the Anderson, drifts are driven in the footwall quartzite. At the Luke and Graveley, however, where the bed dip is greater, the drifts are driven about 5 ft wide in direct contact with the phosphate bed; the drift advance being held up at intervals and the ore slashed down to make a completed drift, 8 to 9 ft wide at the sill. The Luke bed is badly broken by cross faults with offsets ranging from 3 to 20 ft. These faults appear at fairly regular intervals of from 50 to 100 ft and generally displace the bed in the same direction. On encountering each fault, the drift is temporarily stopped, and, as previously stated, the ore in the back is slashed down. Chute locations are selected and sufficient rock is excavated to put in chutes (Fig 6). Unlike the Anderson, timbered manways with ladders and service slides must be carried up each stope owing to the steeper dip of the bed.

STOPING

The area between adjacent faults constitutes a single stope and is mined by a crew of two miners using a 3-in. stoper. Stoping is done in open stopes. The miners drill on planks set up on pipe extension bars called "drill pins" fashioned so as to wedge between the foot and hanging walls. No pillars are required in the stopes and normally no stulls are needed for support as the stopes are of comparatively narrow width and are separated from each other by ground left along the faults.

The ore at the Luke is harder than at the Anderson or Graveley. A set of bits will drill only two or three holes. The rock breaks fairly coarsely and flows by gravity down the bed slope of 45°. In certain sections of the mine the stoping blocks have a decided rake due to the strike of the faults which form the stope boundaries. This cuts down the average dip of the stope to a point where the ore will no longer flow by gravity. On these tilting stopes, the ore is removed by mounting slusher hoists in the manways to drag the ore from the flat parts to where it can drop down to the chutes (Fig 5).

VENTILATION

The mine is largely ventilated by natural means. On the 5300 the incoming fresh air is circulated by electric fans through 16 in. ventilation pipe along the drifts and, where required, in 8 in. lines up the stope manways.

TRANSPORTATION

Luke ore, like that of the Anderson and Graveley, is transferred from the chutes to surface bunkers in 10 car, battery motor-driven trains. From the portal it is hauled on 18 ton trucks seven miles to the railroad cars at Avon.

Graveley Mine

LOCATION AND DEVELOPMENT

The Graveley mine is about two miles northeast of the Luke on the opposite side of the Luke-Graveley syncline (see Fig 1). The strike of the bed, roughly northwest, is the same as at the Luke and Anderson, while the dip is southwest as at the Anderson. The upper part of the Graveley bed has a dip varying from 65° to vertical, while the dip of the beds in the lower level varies from 65° in the southeast to less than 45° in the northwest. Bed widths vary from 3 to 4 ft.

The Graveley ore bed was first developed by an adit driven along it from an outcrop at 5600 ft elevation. From this adit two intermediate levels were opened up at 5750 and 5900 ft elevations to service the higher stopes. A second haulage level was next opened up by drifting in at 5300 ft elevation. The surface slope flattens off below the 5300 level to such an extent that further adits would be impractical.

A winze was collared on the 5300 level, 2000 ft from the portal and sunk in the ore, bed width, and 14 ft along the strike. The winze was put down 550 ft without encountering any appreciable faulting, although the dip of the bed changed from 70° at the collar to 45° at the bottom of the winze. On completion of the winze, a skip was installed with wheels on the sides to allow for maximum capacity in the narrow bed width. The ton skip now in use is capable of hoisting 5000 tons a month, handling ore on two shifts. Owing to the difference of dip in various sections of the winze, a number of 6-in. rollers were put in on both foot and hanging walls. Water was not encountered until the last round and no serious difficulties have arisen in either sinking or operation.

Two levels are being opened up from the winze at 5100 and 4900 ft elevations. Drifting on the 5100 level is being carried on in the same manner as was done in the upper levels and consists of drifting in the footwall and then breaking down the ore from the backs of the drifts in the same sequence as that employed at the Luke. An experiment is being conducted on the lowest, or 4900, level with a different method of drifting. Advance is first made by driving a "scraper drift" in the ore, bed width, and 10 to 14 ft up the dip from the sill of the level. Then the bed is extracted for about the same distance below the sill and finally a drift of regular size is produced by excavating the waste rock from the footwall of the upper part of the slot and dropping it into the space below the sill. Broken rock throughout this operation is handled by slushers. This method has the advantage of eliminating the tramming and hoisting of all waste.

STOPING

Stope preparation, as at the Luke, consists of taking down the ore for a height of 30 ft up the bed and cutting out sufficient footwall to put in a double chute with a manway in the center (Fig 6). These chute cutouts are put in at 50 ft centers.

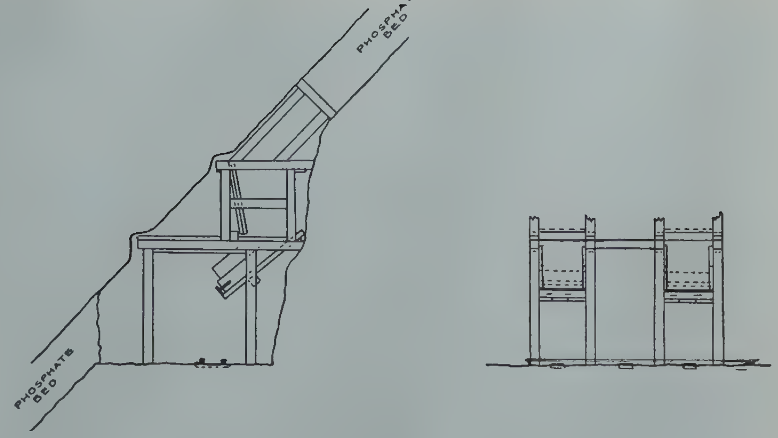

FIG 6—Luke-Graveley stope chutes.

The phosphate bed at the Graveley is fairly regular, the most outstanding fault probably being one which dips northwest and cuts the mine roughly into two halves. This fault is filled with a clay-like material and, although it has a displacement of only 7 ft, it appears to have had a profound effect upon the nature of the ore. To the southeast of this fault the ore is fairly soft and easily drilled, while, for the most part, the ore to the northwest is relatively hard.

Several methods of stoping have been employed at the Graveley. As the bed is quite regular and has a steep dip, particularly in the upper levels, shrinkage stoping was tried out originally. It was soon found that the broken phosphate rock packed in the stope and would not run on even the steepest slopes. A modified shrinkage system was then introduced. This consisted of double chutes at 50 ft centers with a manway in the center. The whole installation was carried up with the stope advance as three timbered compartments, bed width, and each compartment 4 ft wide. The outside compartments were used for ore passage; sufficient rock being carried down them after each blast to facilitate free entrance to the working faces. When the stopes were completed, the broken rock was scraped down into the ore compartments. As the level of broken rock was lowered in the stopes, stulls were placed to support the back. This system required a large outlay for timber to carry the ore compartments and labor to empty the stopes and has, therefore, been abandoned. At present an open-stope method of mining is being successfully used with 8 in. stulls placed at 4 ft centers in rows 6 ft apart. Drilling is carried on from planks laid across the stull rows. Stopes are 100 ft long and are separated by 15 ft pillars. Each stope is serviced by a timbered manway. Broken ore is drawn from two sets of double chutes at 50 ft centers.

TRANSPORTATION

Most of the work is concentrated in the area served by the winze. The ore is trammed by battery haulage to the winze pockets, hoisted to the 5300 and hauled by ore trains to the surface bunkers. From the surface bunkers, it is trucked six miles to the shipping point at Avon.

Phosphate Mining by the Simplot Fertilizer Company near Fort Hall, Idaho

By HEATH B. FOWLER*

Introduction

The surface mining operations of the Simplot Fertilizer Co. are on the Fort Hall Indian Reservation approximately 16 miles east of Fort Hall, Idaho (Fig 1). The Phosphoria formation outcrops in sec. 2, 3, 11, 14, 15, 22 and 27, T 4 S, R 37 E, Boise Meridian. The ownership of these phosphate-bearing lands is divided between individual Indians and the Bannock-Shoshone Tribe, and the royalties are paid to the respective owners on a tonnage basis, as the rock is mined. The geology and mineral resources of the Fort Hall Reservation were studied by the U. S. Geological Survey in 1913 and reported on in 1920.† However, it was not until the winter of 1945 that exploratory work was undertaken by private enterprise. Diamond drilling was begun by the Simplot Fertilizer Co. in April 1946, and stripping operations were started in June. A camp was established near

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† G. R. Mansfield: U. S. Geol. Survey. *Bull.* 713 (1920).

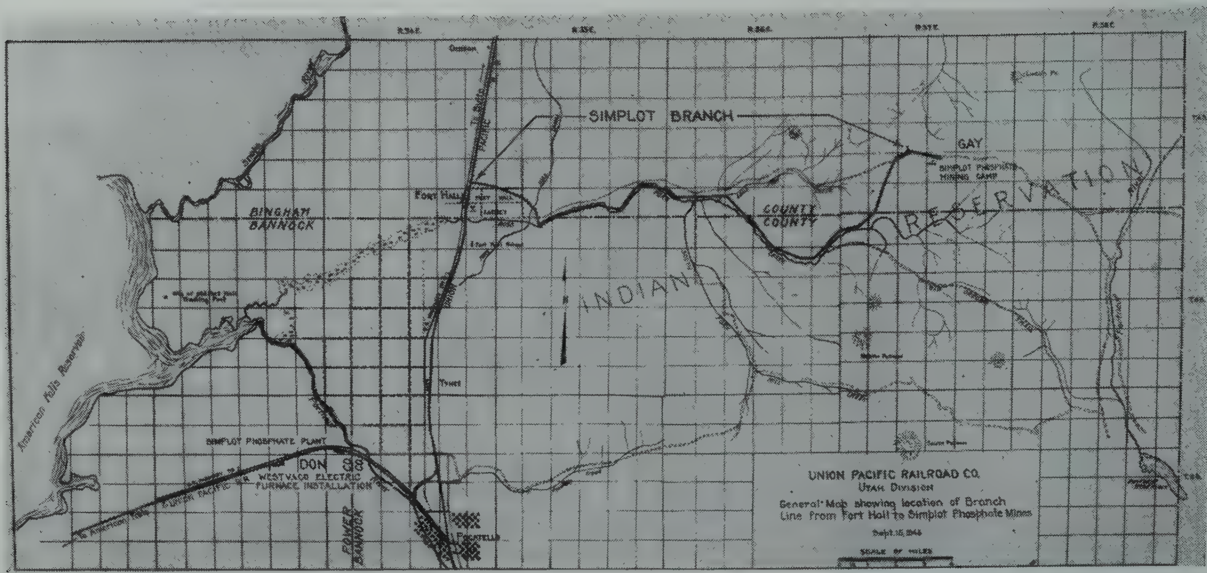

FIG 1—Index map showing location of Simplot phosphate mine at Gay, Idaho, and the Simplot phosphate processing plant at Don, Idaho.

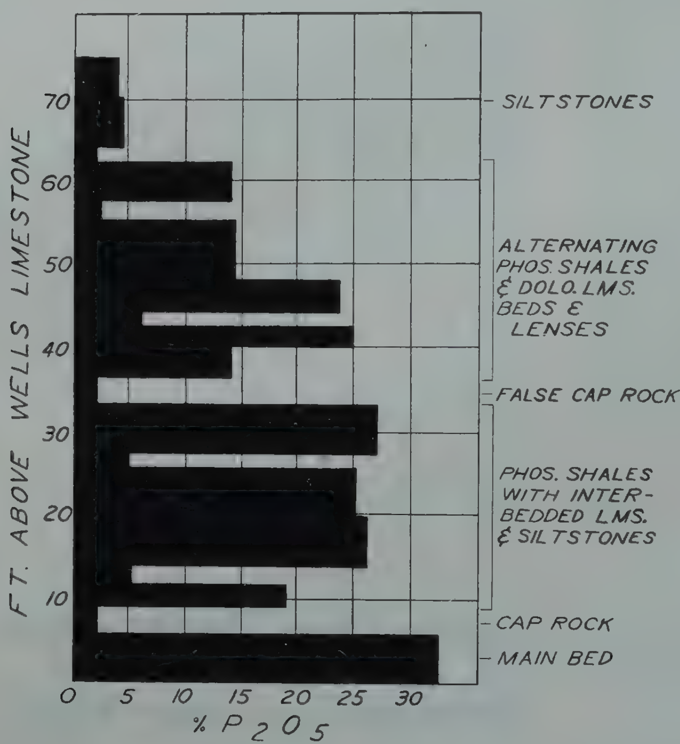

FIG 2—Formation as occurring at Simplot mine showing P_2O_5 content of lower 75 ft of Phosphoria formation.

the south end of the Phosphoria outcrop and 18 miles of oil-surfaced highway was built to provide an outlet to railroad facilities near Fort Hall. A standard gauge spur railroad line, 22 miles in length, was constructed during the summer of 1948 to connect the mine with the Union Pacific Railroad at Fort Hall. The camp buildings include a repair shop, dining hall, change room, bunk houses, offices, and a diamond drill-core library.

Geology

The stratigraphy of the Fort Hall phosphate beds is very similar to that found throughout southeastern Idaho. The general pattern and sequence of the lower 60 to 65 ft of the Phosphoria shale member, as exposed by diamond drilling, stripping and mining operations, is as follows: the bottom 6 ft of oölitic phosphate rock, locally

known as the "main bed," lying immediately above the Wells formation; 2 to 4 ft of limestone, locally known as "cap-rock"; a fairly constant 25 ft thickness of phosphatic shales and another 2 to 4 ft of limestone, which has been termed "false cap rock." Immediately overlying the false cap rock is a series of alternating phosphatic shales and dolomitic limestone beds and lenses approximately 35 ft thick. These are followed by a series of siltstones (Fig 2). The structure of the beds has been determined by a diamond drilling program which has been continuous since the beginning of operations. The major faults and fault zones, which have a definite northeast-southwest pattern, have also been determined and are a major consideration when laying out the stripping and mining panels (Fig 3). The phosphate beds generally dip 6 to 12° in a southeasterly direction and strike from north 5° east to north 45° east, giving assurance of a fairly flat-lying deposit for future strip-mining operations (Fig 4).

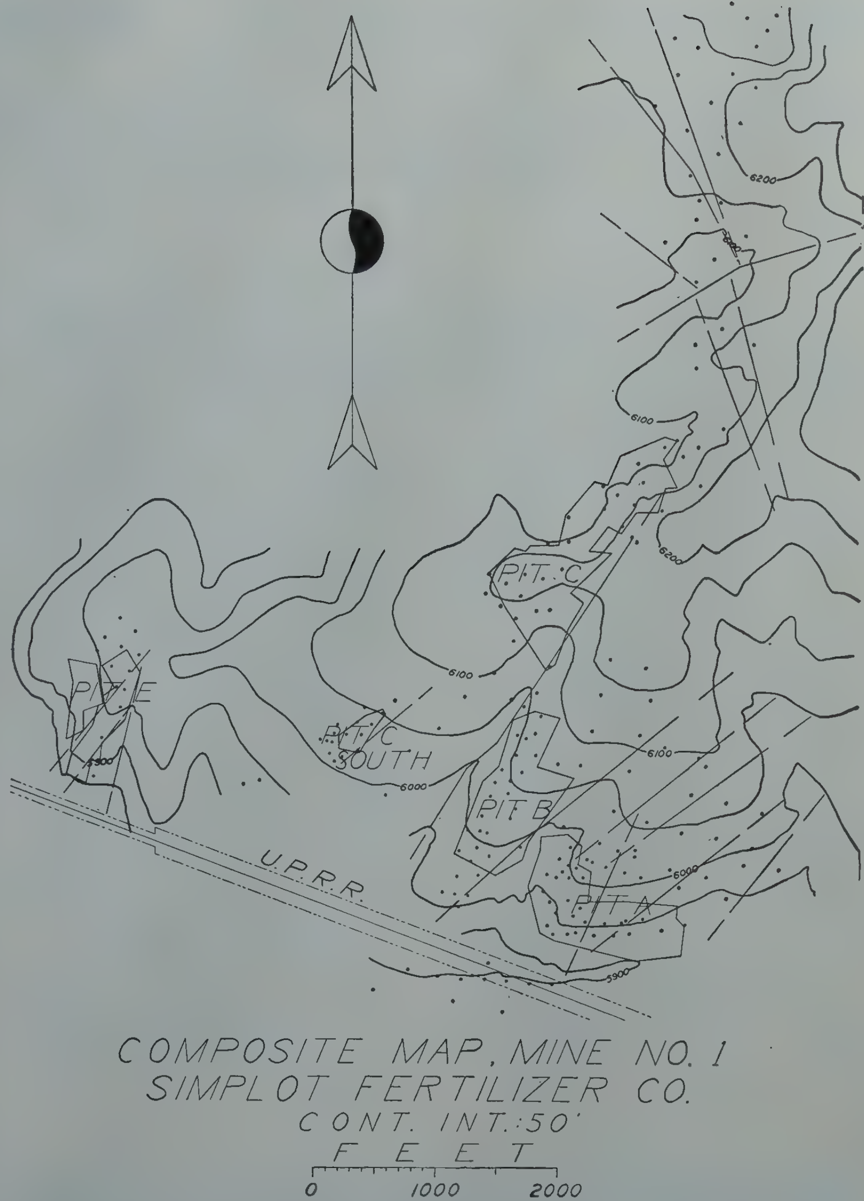

FIG 3—Plan map of south portion, Fort Hall phosphate deposits showing pit areas, diamond drill holes, and major faults.

Production

During the first year of operation at the Fort Hall mine, a total of 53,000 short tons of main bed phosphate rock was mined and hauled to Pocatello, 30 miles southwest, for processing into superphosphate. This first year's production was primarily a large scale sampling and experimental test to determine the feasibility and the economics of mining and processing phosphate rock from a virgin portion of the western phosphate field. During 1947 contracts were made with a department of the U. S. Government to furnish a large tonnage of phosphate rock for export to Japan; this contract, together with an increased expansion in the Simplot Fertilizer Company's processing requirements, necessitated a rapid expansion in the stripping and mining program in order to obtain a production for that year of 500,000 tons of phosphate rock. Additional equipment was procured for all phases of the operation. The truck road to Fort Hall was built up to a standard which would sustain trucks hauling from 15 to 30 ton pay loads. More than 100 trucks were employed in hauling to the railroad at a rate of 5 round trips per shift and 2 shifts daily. It was necessary to construct suitable detours which would stand up under such traffic while the actual road grading and black topping was in progress. The completion of the government contracts in early 1948 brought the mining program back on a more even and steady working basis and saw the beginning of selective mining and stockpiling of phosphatic shales for electric furnace feed.

The total production of phosphate rock as of Dec. 1, 1948, at the Fort Hall mine has been 786,000 tons mined and shipped. In order to uncover this amount of rock, it has been necessary to remove 900,000 yards of top soil, 155,000 yards of limestone cap rock, and 1,800,000 yards of lower-grade phosphatic shales. The approximate stripping ratio of ore to waste was one to five.

In stripping the overburden, very little blasting is required to prepare the material for shovel loading; however, a few plugs are drilled occasionally to break some large blocks of limestone. The experience of the past two years has shown that the phosphatic shales and siltstones stand well even when the pit walls are nearly vertical and up to 100 ft in height.

Diamond Drilling

It is necessary to explore the Phosphoria formation with a methodical

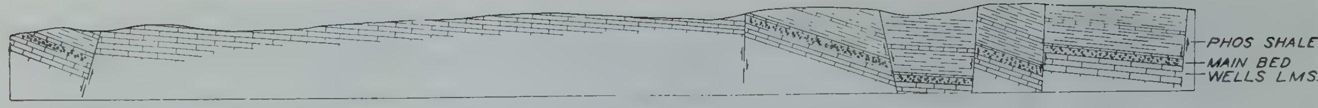

FIG 4—Sketch of typical cross section as determined by diamond drilling and pit exposures, showing fault blocks common at Simplot mine. Scale approximately 1 in. equals 100 ft. (Reduced approximately 50 pct in reproduction.)

pattern of diamond drill holes before any mining plan can be formulated because of the frequent faulting and the presence of top-soil overburden. Through experimentation in the field, it was found most practical and economical to use a 150 ft grid pattern with each corner being a diamond drill-hole site. The grids are laid out on the ground by transit and stadia and are parallel to the strike of the beds. All diamond drilling is now done by a company-owned drill. In the past this was augmented by contract drillers employing up to five machines. Drills are worked two shifts a day with a driller and a helper as one crew. Each hole is drilled BX to set the stand pipe in solid rock and then reduced to AX for the remainder of the hole. At the completion of each run, the holes are reamed and the casing extended to the bottom. This procedure minimizes caving of the higher beds and subsequent contamination of the sludge. No hole was completed until the underlying Wells formation was recognized. The deepest holes to date are 250 ft with the average being 72 ft. A total of 18,000 ft has been drilled.

It is very difficult to obtain a good core recovery in drilling the shale formation by ordinary drilling practices; hence, ways and means have been found by experience to double the original recovery percentage. Even so, there is room for additional improvement. The principal practices which have produced the most satisfactory results are to ream and case to the bottom of the hole at the end of each run, the use of the "face-discharge" bit while drilling soft phosphatic members, short drill runs, and strict vigilance in controlling the proper water supply at the bit, this control being determined by the hardness of the formation.

During the past year a bonus system based on core recovery was set up for the drill crews. This system has resulted in a marked increased of core recovery. All drilling is under the supervision of an engineer to ensure proper casing and sampling of each hole.

In order to overcome the discrepancies and misconceptions resulting from a lack of actual core, great emphasis has been placed on the recovery and sampling of sludges.

As mentioned previously, care is taken to prevent caving of the holes by constant reaming and casing. Hence, the sludge from each drill run is fairly representative of the footage drilled.

This sludge is recovered in sludge boxes especially made for this purpose, having large settling areas and several baffles with a small amount of fall. These permit the recovery of all material except that which is subject to sliming. These slimes are discarded and an allowance made to compensate in the subsequent sampling results.

As an added precaution against salting of the sample, water, even though rather scarce in the area, is piped and pumped up to a distance of two miles in order to prevent having to recirculate it.

The drilling program, based on the 150 ft grid pattern, brings to light the location of the faulted blocks. In several instances, holes have been collared and drilled into the fault zones and very little stratigraphic information was obtained. Whenever such a condition is encountered, the location is moved a few feet and another hole put down.

The drill rigs used by the company and the contractors were Duralite Model 1 S surface drills with a gear drive screw feed. These machines have a capacity of 800 ft with *A* rods.

Future plans call for the addition of a hydraulic feed drill rig to further improve core recovery.

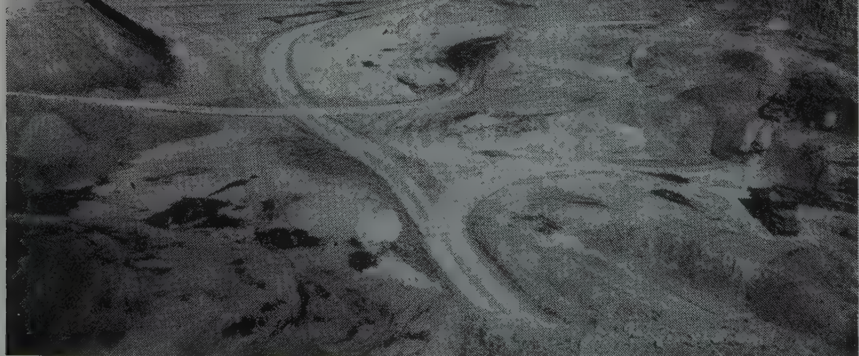

FIG 5—View of strip mining operation from removing top soil by caterpillar and carryalls in background, to mining high grade rock in the foreground. Euclid dump truck in left center is stock piling furnace feed shale.

Stripping

Various methods of stripping have been tried and compared since operations began, and various time studies

and efficiency records have been kept on each piece of equipment used. These tests conducted during the last three years have resulted in the choice of the following equipment: 3 D-8 caterpillar tractors with caterpillar Model 80 scrapers, capacity 15 to 18 yards; 2 D-8 caterpillar tractors with bulldozers; 1 HD-19 Allis Chalmers tractor and bulldozer or push bell; 2 80D Northwest 2½ yard shovels; 1 37B Bucyrus Erie 1½ yard shovel; 1 Model 55 Bay City 1 yard shovel; 1 Model 25 Northwest ¾ yard shovel; 7 Model 49FD Euclid end dump trucks, 11 yards capacity; 5 Model 548 Kenworth dump trucks, 12 yard capacity; 1 Model 12 caterpillar patrol; 1 Model 212 caterpillar patrol; and 2 4500 gal water tank trucks.

Both the bulldozers and the scrapers are used in the removal of top soil. The economic maximum haul is 500 ft with the scrapers and 300 ft with the bulldozers. The 2½ yard shovels are used in conjunction with the pit trucks in removal of the phosphatic shales. This size shovel is particularly adaptable to the pit operations because of its mobility, loading speed, and capacity. The smaller shovels, ¾ to 1½ yard have proved to be better adapted to mining the high-grade phosphate rock. Their superiority is primarily due to the greater selectivity with which the operator can control the smaller shovels and thus obtain a much cleaner product for shipment.

Model 12 and 212 patrols are used to keep pit roads in good condition. This practice, in conjunction with daily sprinkling of the roads by water trucks, increases the number of loads per truck shift by 20 pct and decreases repair costs on equipment moving over dustless roads. Two D-8's with bulldozers are in the pits at all times, removing overburden or cleaning the main bed preparatory to actual mining and stripping.

Mining Plan

Prior to any strip-mining operations, mining panels are staked out on the ground as indicated by the assembled geological data. The panel size is controlled by fault blocks, depth of overburden, and the topography as it affects the location of spoils areas. After a panel is laid out, the first stage of mining involves the removal of all top soil from the area. This is accomplished by caterpillars and carryalls scraping the top soil down the natural slope of the ridges.

Large-scale sampling and mining experiments on the phosphatic shale beds have shown that these beds range considerably in the geologic column, but several beds of considerable thickness have a P_2O_5 content that makes them suitable for electric furnace feed. As a result of these tests all phosphatic shale members overlying the main bed are mined selectively, under close supervision, guided by a sampling program, which embraces sampling of the various beds from the time they are first cored in the drilling until they have been mined and stockpiled.

The Westvaco Chemical Division of the Food Machinery and Chemical Corp. is constructing a 13,000 kw electric furnace near Pocatello, Idaho, for the production of elemental phosphorous. Operation of the furnace is scheduled to begin in the spring of 1949. Phosphatic shales for the furnace feed will be obtained from the Fort Hall mine. Until the furnace is put in operation all shales mined and earmarked for the furnace are stockpiled in convenient areas near the mining pits, and daily records are kept of tonnage and P_2O_5 content.

Selective strip mining in a flat-bedded sedimentary deposit requires the use of benching. The height and area of each bench is determined by assay boundaries. These resulting benches, although appearing in a haphazard array, do have a continuity which is recognized when the whole geologic picture is known. It is the physical and chemical lensing of individual shale beds which controls the P_2O_5 content and hence the various boundaries and levels (Fig 5).

All shales which fail to contain the proper P_2O_5 for the furnace are piled conveniently and separately as a unit however, all top soil, limestone, and siltstones of very low P_2O_5 content are wasted and kept separate as another unit. Once the top soil, shales, and limestone are removed, the high-grade bed is ready for mining.

Inasmuch as it is necessary to deliver to the Simplot Fertilizer processing plant at Pocatello a rock containing a minimum of 31.5 pct P_2O_5, and as the rock in place contains from 31.5 to 33.5 pct P_2O_5, great precaution against dilution must be exercised during its removal by power shovel.

After removal of the overlying limestone cap rock, the bed is cleaned very lightly by a dozer. The most experienced men are used on this job.

The actual removal and loading of the rock by shovel is fairly simple as the footwall of the bed is designated by an abrupt change of color. The phosphate bed is black to dark gray, whereas the footwall is a red-brown siltstone. However, there is a problem of small "operational faults" which requires the skill of an experienced operator. These faults have throws of from 1 to 6 or 7 ft and are usually not evident in diamond drill-hole cross sections. They are later picked up during mining operations.

The main bed is soft and friable and is shoveled directly into pit trucks and hauled to the railroad for loading into cars. The present loading facilities consist of a ¾ yard shovel dumping through a small hopper onto a 30 in. conveyor belt, inclined for direct discharge into the cars. Although this arrangement loads 70 tons per car in 10 min, it is necessary to rehandle the rock during the loading operation. Proposals are now being considered to combine crushing, automatic sampling, and loading of the cars in one operation.

Although the equipment now in operation at the mine is the most suitable for selective mining of the lower phosphatic beds and the requirement for equipment flexibility, future mining will entail greater depths of overburden. A large proportion of this future overburden will not be amenable to selective strip mining but will be removed as units up to 75 ft in depth. These large yardages of waste material, low in P_2O_5 content, necessitate overburden removal at a low cost. Comparison studies are now being made on the larger types of earth-moving equipment.

Although Idaho has a vast reserve of phosphate rock, there is a very limited amount structurally suitable for strip-mining operations. Present day high costs and the market price of the rock, which is lagging in comparison, coupled with complex structural and stratigraphic problems require the utmost vigilance on all phases and details of a strip mining operation.

Safety in Mining at the Andes Copper Mining Company's Property, Potrerillos, Chile

By C. M. BRINCKERHOFF,* Member AIME

Safety work in mining at the Andes Copper Mining Company, Potrerillos, Chile, is divided into three parts: (1) accident prevention, (2) fire prevention and protection, and (3) silicosis prevention and control.

In the company's operations a steady campaign has been waged to reduce accidents and silicosis, and to prevent fires. In the following pages the safety record for the past twelve years will be examined and the methods explained by which the accident rate was reduced.

Table 1 contains yearly data pertaining to the scale of the mining operations and the accident record. Block caving is used for all ore production and the tonnage mined has averaged 30,000 short tons per day for the past seven years.

Table 2 classifies 44 fatal accidents which have occurred since 1935. The leading causes of accidental death are transportation of ore and use of explosives, followed in order by persons falling, falls of ground, and by asphyxiation.

In the case of serious accidents, however, the greatest cause is falls of ground, followed, in turn, by ore transportation, persons falling, explosives, and asphyxiation.

Accident Prevention

The improvement of the safety record was obtained by a combination of many factors. In surveying the results at Andes for the past twelve years, it is possible to select some features of the present operation which have contributed to the accident record. The most important of these are: (1) good staff planning, (2) good equipment, (3) an alert safety organization, and (4) education and selection of the workmen.

GOOD STAFF PLANNING

As a result of good staff planning, the mine has good working conditions, namely, good ventilation, moderate temperatures and humidity, and good lighting. These factors not only reduce accidents but also improve efficiency.

In haulage work, accidents have been reduced by having traffic move in a forward direction only. The latter makes possible a one-man train crew. Trains are centrally dispatched from loading drifts which are interconnected by signal lights, so that when a train in a lateral drift is given permission to move into a main line drift, a red light signals in all other lateral drifts in that area. Good track is maintained, and safety niches are required at all switch throws. Since safety niches were provided there have been no fatal or serious accidents at such points. All main line haulage drifts are illuminated by electric lights. Haulage drifts are kept clean. Adequate clearance has been provided between cars and between cars and the sides of the haulage drifts. The 4½ ton Granby type cars are equipped with roller bearings, and the 10 ton locomotives with trolley shoes instead of wheels. These two features are responsible for improved ore haulage conditions and contribute to greater efficiency and safety.

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Table 1 . . . Scale of Mining Operations and Accident Record

Year	1936	1937	1938	1939	1940	1941	1942	1943	1944	1945	1946	1947
Mining Operations												
Metric tons ore mined	1,933,509	5,245,505	5,188,988	5,035,391	6,964,376	8,644,888	9,167,599	10,151,507	8,784,930	8,225,716	8,965,783	9,684,259
Lineal meters development work driven	10,350	26,460	36,565	28,736	31,109	34,568	45,023	52,266	49,166	90,192	35,014	40,525
Area undercut square meters	6,163	20,602	35,382	26,539	26,222	36,673	39,894	48,252	49,952	39,786	31,819	26,234
Accident Record per 10,000 Shifts												
Slight lost time	4.30	7.61	6.96	4.09	3.73	2.44	1.105	2.079	2.225	1.144	1.517	1.660
Serious	0.90	0.92	0.88	0.58	0.37	0.43	0.414	0.410	0.178	0.254	0.269	0.328
Fatal	0.07	0.15	0.02	0.16	0.07	0.06	0.017	0.091	0.107	0.106	0.049	0.044
Total	5.27	8.68	7.86	4.83	4.17	2.93	1.536	2.580	2.510	1.504	1.835	2.032

GOOD EQUIPMENT

In providing equipment for the miner at Potrerillos, the underlying idea has been that the best equipment possible will result in fewer men underground, a lesser risk, and greater efficiency.

The best drilling equipment available has been provided the machine man. Only electric blasting is used when blasting is necessary in ore drawing and car loading. Haulage and grizzly levels are connected by telephone for communication in such blasting work. All undercuts are blasted with Primacord, and in raise and drift work a minimum fuse length of 8 ft is used. Hot-wire lighters are used for "spitting."

Protective clothing is provided free to the miner. This includes hard-toe shoes or boots, gloves, and hard hats. The "wet" driller also receives a waterproof coat or jacket.

Another prominent cause of accidents is "falls of persons." This has occurred in ore-pass work or in work in surface pits. In this type of work the use of safety belts and ropes is mandatory and accidents are practically always the result of disobeying orders.

SAFETY ORGANIZATION

The safety department comprises a safety engineer and twelve experienced men. Ten of the latter go underground with the working shifts, inspect working faces, help the shift boss in the section to which he is assigned, especially at blasting time. The safety man is distinguished by a white helmet and is always present at the loading of undercuts and the tying-in process with Primacord. He is usually the first at accidents and summons what help may be necessary. As the hospital is close to the shaft collar, a doctor or male nurse is called underground if such services are required. A minimum of first aid is administered underground because of the proximity of the hospital.

In addition to their underground duties, the members of the safety department give talks on accident prevention at the Syndicate Hall. The safety department maintains close contact with the syndicate or union, and endeavors to see that the members are familiarized with all safety posters and circulars. At present a syndicate director is a member of the underground safety organization—a circumstance which enables the company to reach the men more easily in the safety campaign.

The safety department screens all men before they are hired and also checks the accident records, so that a man with a poor record can be eliminated from the underground force.

The safety department is also in charge of all blasting cap testing, fuse cutting and capping, and fuse and dynamite tests. All electric caps are tested for "shorts" and sensitivity in an apparatus designed by Frank W. Bell, superintendent of the electrical department. Fifty caps are tested in one operation by placing them in individual recesses, connecting wires to separate binding posts and passing 240 milliampers through each cap by a push-button control, (see Fig 1). The tested caps are then marked, repacked in boxes, and made available to the mine. About one cap in 5000 is found defective.

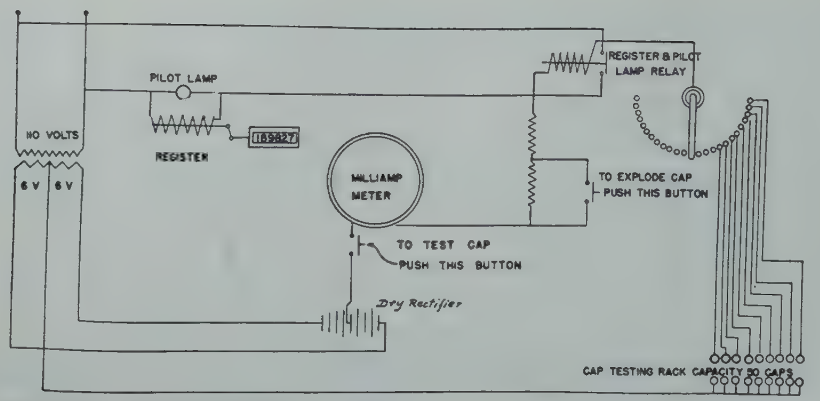
The safety department is in charge of ventilation, dust sampling and counting, dust control, fire prevention, and rescue apparatus and training. The underground members of the department participate in a monthly bonus and a yearly prize based on their record and contribution to the mine safety. Awards are made by a committee composed of three members of the syndicate and three of the mine staff, presided over by the mine superintendent. The main purpose of the bonus and prize system is to stimulate interest in accident prevention.

EDUCATION AND SELECTION OF WORKMEN

In the previous paragraphs the work of the safety department was mentioned in connection with the education and selection of workmen. Close cooperation is maintained between the mine staff, the safety organization, the medical department and the syndicate. Recommendations by the doctors have an important bearing on whether a man is considered suitable for underground work, or physically capable of doing the work expected of him. The mine superintendent and safety engineer discuss all accidents of a serious nature with the syndicate directors, with a view to avoiding a repetition of the accident, and to agree on proper punishment for safety rule violations.

Table 2 . . . Classification of Accidents

	Falls of Ground	Falls of Persons	Ore Transportation	Use of Explosives	Asphyxiation	Miscellaneous	Total
Fatal Accidents							
1936 to 1947, inclusive	4	6	11	10	3	10	44
Serious Accidents							
1936 to 1947, inclusive	80	19	26	19	1	112	257



NOTE: CAP SHOULD HOLD UP TO 240 MILLEAMPERES OR 600 ON SCALE.
CAPS EXPLODE AT 360 MILLEAMPERES OR 900 ON SCALE.
2.5 ON SCALE = MILLIAMPERES.

FIG 1—Electric cap tester at the Andes Copper Mining Co., designed by W. F. Bell.

Fire Prevention and Protection

The safety department at Potrerillos is in charge of all fire fighting equipment and training. At present there are on hand five 2 hr McCaa and five Chemox machines. Five additional Chemox apparatus are on order. There are 10 five-man teams trained in rescue work and the use of fire fighting equipment.

For fire protection underground, the following measures have been adopted:

1. There are no buildings within 75 ft of any shaft collar or adit.
2. All shaft collars and adits have doors which can be closed manually. This is for smoke control in case of a surface fire.
3. All levels have doors close to the shafts, so that the shaft can be isolated. These doors are of steel plate construction and are of a sliding type, fitting into a slot in the side of the drift and hung from wheels which ride on a steel angle bar across the top of the drift.
4. Every shaft station is equipped with a barrel of water and a bucket.
5. All cages carry a special water hose which can be connected at each station to a special water line painted red.
6. No smoking is allowed on cages or in shaft stations.
7. No underground forges or tool sharpening are allowed.
8. All locomotive repair shops have fire extinguishers, a box of sand, and water hose and connections.
9. Electrical cables pass from the surface to all levels through raw or concreted raises.
10. The use of timber underground is kept at a minimum—at present 0.14 board foot per ton of ore. This low figure has been attained by close supervision and by the development of improved methods.
11. Special trash collections are made daily from underground lunch rooms, powder magazines, and the fuse-cutting room.
12. All areas of the mine are connected to the surface by telephone and similarly interconnected underground.
13. Ventilation plans are kept up to date.
14. Each shift boss has a copy of the ventilation plans, showing all exits and containing instructions as to the location and direction of fresh air currents in case of an emergency.
15. In case of an emergency, changes in the ventilation system may be made only by authority of the mine superintendent.

Silicosis Prevention and Control

The prevention and control of silicosis at Potrerillos has been a particularly difficult problem, due to the dryness of the climate. The small mines in the north of Chile are dry and no water is available for wet drilling. Applicants for work at Potrerillos who have worked as long as three years in the small, dry mines are usually rejected by the company doctors on the basis of their lung X-rays.

Encouraging results have been obtained by dust control measures, and by making lung examinations of applicants for work. Results of examinations for the year 1947 showed that the lung condition remained unchanged in 91 pct of the mine personnel.

The points which have contributed most to reducing the silicosis hazard and improving working conditions are:

1. Wet drilling throughout the mine.
2. Good ventilation. 500,000 cu ft of air per minute are exhausted from the mine.
3. An integrated ventilation system for the mine which is constantly being brought up to date. The underlying idea in the ventilation is to remove dirty air from mine and not to allow it to get into circulation again.
4. Wetting down haulage drifts at night.
5. Water sprays at ore dumping stations.
6. Dust control doors at ore-pass connections and wherever they are required.
7. The use of "Comfo" type respirators. In the past eleven years 13,802 were issued to the men. The wide use of the respirator has been developed with the cooperation of the syndicate and through education of the men by the staff.
8. All men are examined yearly by X-ray for silicosis or other lung disorders. The recommendation of the examining doctor is followed in cases where lung markings are indicative of trouble.
9. An alert cooperative medical staff headed by Doctor Jorge Campino.

Summary

The safety program of a mining operation is always an unfinished job. New and untrained personnel are being hired regularly. New equipment brings unforeseen hazards and problems. However, any record can be improved if an intelligent and humane attitude is taken by management and staff. Many of the safety practices now in force were started by I. L. Greninger when he was Mine Superintendent and, later, General Manager of the Andes Copper Mining Co. At present, R. L. Tobie, Mine Superintendent, Harold E. Robbins, Assistant Mine Superintendent, and Juan Abarca, Safety Engineer, are carrying on with policies and programs initiated by their predecessors.

Studies on the Activation of Quartz with Calcium Ion

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That calcium will activate quartz for flotation with anionic collectors such as soaps has been known for a number of years,[1,2,3] and the method has been applied to the concentration of various iron ores[4,5,6] as well as to other materials. Clemmer gives the permissible pH range for the activation of quartz by calcium as between 9.5 and 12.5 with an optimum value of 11.0, and Hertzog[7] has determined the calcium abstraction of iron ores and of Ottawa sand, but otherwise little quantitative data are available.

The purpose of the present investigation was to determine those conditions under which dilute solutions of calcium chloride will activate quartz for anionic flotation and to determine the amount of calcium ion adsorbed by quartz. Those variables, the influence of which was primary, such as calcium, sodium, and hydrogen ion concentrations, were studied. Other factors which influenced the results are discussed in the description of the experiments.

Three different experimental methods were used to determine the activation of the quartz. The first method was to treat a sample of ground, deslimed quartz with solutions of calcium chloride and sodium hydroxide, and to determine the calcium adsorbed by chemical analysis of the solution. Advantages are that a direct value for the amount of calcium adsorbed is obtained, and introduction of a collector which may affect the adsorption is avoided. The difficulty is to determine the extremely small differences in the calcium concentrations before and after the abstraction. The second method was to determine the activation of the quartz by observing the contact angle obtained on pressing an air bubble against the polished surface of a piece of quartz immersed in a solution of calcium chloride, sodium hydroxide, and sodium oleate. This is the well-known "contact angle" method. The third method was to determine those conditions under which an air bubble will pick up quartz particles immersed in a solution of the desired reagents.

Disadvantages common to the second and third methods are that introduction of a collector is necessary to obtain bubble contact with the mineral; very small quantities of contaminant can give erroneous results; and no information is obtainable concerning the amount of calcium adsorbed. Advantages are that tedious analyses, which at best give approximate results, are avoided, and the conditions, especially for the third method, closely simulate those of actual flotation. Although the most consistent results were obtained with the third method, this paper describes the techniques used and the results obtained from each of the three methods.

Distilled water and analytical grade reagents were used throughout except where otherwise stated. Between tests all glass articles were cleaned with a chromic-sulphuric acid mixture and sodium hydroxide.

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[1] References are at the end of the paper.

Abstraction Method

The raw material for the tests was Ottawa sand sized between 14 and 28 mesh. Batches of 500 g of this sand were dry ground for 1 hr in an Abbe porcelain mill with 2700 g of quartz pebbles. The batches were split down to samples weighing approximately 30 g, and the weight of each sample was adjusted to exactly this value.

Thorough wetting and desliming of each sample was effected in a Schoene funnel. This was found necessary for three reasons: first, the grinding produced some very fine porcelain that was capable of disturbing the adsorption equilibrium; second, it was impossible to separate the adsorbtion solution from samples containing slimes without first filtering, and tests showed that the filter materials can adsorb calcium from solution; third, the surface area of samples containing slimes is difficult to measure or even to calculate.

The weights of the deslimed samples prepared from the same batch of ground sand ranged from 22.9 to 23.2 g, indicating that the variation in size distribution between samples was slight. The wet, deslimed samples were transferred to 125 ml Pyrex Erlenmeyer flasks and the tests were carried out in these.

The samples were first washed three times with distilled water, then treated 1 hr on a mixing wheel with $0.1N$ hydrochloric acid to dissolve any activating ions present. The samples were then washed five times with distilled water and stored in distilled water for about 6 hr before being used.

For reasons subsequently explained, in some cases the samples were treated with dilute sodium hydroxide solution following the acid treatment, the pH of the wash water being adjusted with sodium hydroxide to that value desired in the adsorption test.

A sizing analysis of a representative sample of the deslimed material gave the results shown in Table 1.

Table 1 . . . Sizing Analysis of Deslimed Quartz

Mesh	Weight Per Cent
+35	0.5
−35 + 48	4.5
−48 + 65	12.0
−65 + 100	18.0
−100 + 150	16.0
−150 + 200	13.0
−200	36.0
	100.0

From Stokes' law and the known water velocity in the desliming funnel, the lower size limit of the grains in the samples was close to 10 micron This checked well with microscopic measurements of the lower size limit for the deslimed sample and with the upper size limit for the slime. Under the microscope the grains in the deslimed fraction appeared to be free from adhering slimes.

Surface area determinations were made on samples from the different grinding batches used in the adsorption tests, using the air permeability method.[8] The results were 440, 459, and 550 sq cm per gram of deslimed quartz. This method of surface determination gives only the surface area of the particles, whereas adsorption measurements give larger values because of the presence of cracks and pores. The ratio of surface as given by gas adsorption and air permeability methods on identical quartz samples is 1.8, and applying this correction to the average of the three values given above gives an average surface area of 870 sq cm per gram of quartz. That this cannot be too much in error is supported by calculations made from the sizing analysis previously given. When Ottawa sand was ground under identical conditions and the minus 200 mesh fraction was sized by sedimentation, it was found that a plot of the logarithm of size against logarithm of weight gave a straight line.[9] Using the appropriate correction factors given by Gross and Zimmerley[10] and by Gaudin and Rizo-Patrón,[11] the computed surface area of the material under investigation was 760 sq cm per gram.

Immediately before making an adsorption test, the wash water was changed and the volume adjusted so that it would be exactly 75 ml after all reagents had been added. Calcium chloride was added as a 3 g per liter solution, and sodium hydroxide as a 0.1 or $0.01N$ solution, depending upon the pH desired.

The flasks containing the quartz and reagents were revolved for the desired adsorption time on a mixing wheel at 100 rpm. The quartz then was allowed to settle for 3 min, and 50 ml of the supernatant liquid was carefully drawn off with a pipette. This portion was used for the calcium determinations, the remainder being used for a colorimetric pH determination. Blank tests were run using the same solutions but with no quartz present, and the quantity of calcium adsorbed was taken as the difference between the tests with and without quartz.

ANALYTICAL PROCEDURE

The 50 ml sample was diluted to 100 ml, two drops of bromthymol blue were added, and the solution neutralized with $1N$ hydrochloric acid plus 10 ml in excess. Then 10 ml of an 80 g per liter solution of oxalic acid was added, the solution was heated to boiling and neutralized by slowly adding dilute (1:4) ammonium hydroxide and then a few drops extra. The calcium oxalate was allowed to precipitate overnight and was filtered in a Gooch filter crucible with a filter bed of short fibered asbestos which had been leached in hot sulphuric acid (1:10) and hot ammonium hydroxide (1:4). The precipitate and beaker were washed four times with hot dilute ammonium hydroxide (1:1000). The calcium oxalate then was dissolved with 100 ml hot sulphuric acid that was poured through the filter crucible. The solution was collected in the same beaker in which the precipitation had been carried out to avoid the loss of calcium oxalate adhering to the beaker walls. The solution was heated to boiling and titrated hot with $0.05N$ potassium permanganate.

This method gave good correlation between parallel determinations. After some practice it was possible to determine calcium in amounts down to 5 mg with an accuracy of plus or minus 0.1 mg. For smaller quantities of calcium the accuracy decreased rapidly, probably because of incomplete precipitation of calcium oxalate from such dilute solutions.

As the amount of calcium adsorbed is determined as the difference between two independent analyses, the accuracy of the values given is plus or minus 0.2 mg for the whole sample, or plus or minus 0.01 mg of calcium adsorbed per gram of quartz. With microanalytical equipment for calcium determination, as described by Kirk and Moberg,[12] it should be possible to increase the accuracy substantially.

In the first determinations filter paper was used for separating the calcium oxalate, but it was found impossible to correlate parallel tests, and the determined values of adsorbed calcium were always too large. Blank tests with no calcium present demonstrated that oxalate ion was adsorbed in the filter paper from alkaline solutions, and that even excessive washing failed to remove it. However, it dissolved easily in the sulphuric acid used for dissolving the precipitate.

RESULTS

In the initial investigations only quartz samples were used which had been leached in hydrochloric acid alone. The results of the tests with these samples differed greatly from those obtained from later tests where samples treated with both hydrochloric acid and sodium hydroxide were used. The values for abstraction of calcium ions were larger and increased rapidly with the pH and increase in treatment time, contrasting with the later tests where the adsorption values were low and

rather constant for a wide pH range. The difference was greatest at high pH and characteristically the adsorption solutions were cloudy after the tests on quartz which had been pretreated with both acid and alkali. As the cloudy solution gave a Tyndall cone, it appears that colloids were dissolved in the initial tests, and the most probable explanation is that some residual slime quartz dissolved in the alkaline abstraction solution, giving a silica sol which then precipitated some of the calcium as calcium silicate. Kellogg and Vasquez-Rosas[13] claim that finely ground quartz is rather soluble in dilute sodium hydroxide solutions, and they give a solubility of 64.4 mg quartz per liter for 0.001N sodium hydroxide. It is obvious that even far smaller quantities of dissolved silica will affect the abstraction values of calcium very substantially.

To avoid this difficulty, the quartz samples, after acid treatment, were treated for 16 hr with sodium hydroxide solution at a pH of 12.1 and then were washed with sodium hydroxide solution of the pH desired during the adsorption tests, thus removing colloidal silica or any soluble silicates arising from residual quartz slimes. The results obtained from a 1 hr adsorption period using various pH values and two calcium concentrations are given in Table 2.

The values are given as milligrams of calcium adsorbed per gram of quartz. The accuracy of the determinations is plus or minus 0.01 mg calcium per gram of quartz, or plus or minus 33 pct for the largest value found. However, the results show the trend to be a slow increase in adsorption to a pH of about 11.0 and a fairly constant value thereafter in contrast with the rapid increase in adsorption above 11.8 found with quartz which had not been pretreated with sodium hydroxide. This indicates that silica has not dissolved in such amount as to disturb the results.

DISCUSSION

To form a close-packed monolayer of calcium when an ionic radius of 1.0×10^{-8} cm is assumed, about 50 micromols of calcium per square meter of surface is required, but it is more likely that calcium ions are adsorbed to the oxygen points in the quartz lattice. Gaudin and Rizo-Patrón[11] have calculated the number of such points from structural data for quartz, and have found a value of 4.27×10^{18} points per square meter, corresponding to an adsorbing capacity of 7 micromols per square meter. The adsorption which they found experimentally for barium ions was 10 micromols per square meter, and to explain this excess they assume that the barium ions also are adsorbed to the hydroxyl ions on the surface by exchange between barium ions in the solution and the hydrogen in the hydroxyl ions. In this way the adsorption capacity of the quartz surface under optimum conditions should double to 14 micromols per square meter.

Taking the mean of the five values found for calcium for a pH of 10.8 or higher (Table 1) it is found that 0.024 mg of calcium is adsorbed per gram of quartz. Using the average surface of 870 sq cm per gram of quartz as previously determined, this gives an adsorption of 7.1 micromols of calcium per square meter of surface, very close to the theoretically derived value.

Contact Angle Method

Several specimens of water-clear Montana vein quartz were selected for the contact angle tests. One face of each specimen was ground plane with carborundum on a glass plate. The specimens were mounted in Lucite by a standard briquetting procedure, and the exposed quartz surfaces were ground with 600 carborundum and finally polished with levigated alumina and distilled water on a polishing wheel.

To obtain fresh surfaces for the captive bubble tests, the following procedure was adopted: the specimens were wiped with a cloth moistened with sodium hydroxide solution at a pH of 12.0, polished by hand on a clean cotton pad with alumina and distilled water for some minutes, washed with distilled water and repolished on a new pad with fresh alumina and distilled water. After a thorough washing they were mounted in the bubble cell for the test run. All quartz and briquette surfaces were kept wet throughout.

Specimens treated in this way gave no contact with the air bubble when tested in distilled water, or in calcium chloride and sodium hydroxide solution. Some tests were made in which the specimens were given a final polish with alumina and sodium hydroxide at a pH of 12.0. These tests gave distinct contact angles in sodium oleate solutions for a pH ranging from 7.0 to 12.0, and the explanation is probably that some alumina dissolved as sodium aluminate and activated the quartz during the polishing. Tests run in the same way, except that minus 5 micron quartz was substituted for the alumina, gave only cling contact in the same solutions. All results described later were obtained using alumina and distilled water to avoid the activation described above.

Tests were made in the usual fashion except that the polished surface was turned down to prevent any precipitates formed from solution from accumulating on the quartz. This required that the bubble, ranging from 2 to 5 mm in diameter, be brought up to the polished surface. The values of the contact angles obtained were estimated, with an accuracy of plus or minus 5°. Solution temperature ranged between 20 and 25°C.

TEST METHOD

The specimen was suspended in the bubble cell in distilled water. Calcium chloride and sodium oleate solutions were added in that sequence, with thorough stirring between each addition. After a lapse of 3 min the specimen was tested by pressing a bubble against it for 100 sec. This was repeated on at least three places on the surface. The value of the contact angle, if any, was noted. When contact occurred, the time necessary to give contact (induction time) was determined by pressing the bubble against the surface, first for only 1 sec, and then for increasing time until a contact was obtained. The contact angle obtained in this way was also noted, and will be called the initial contact angle.

The next step was to add a small amount of sodium hydroxide and to repeat the whole procedure. Then fresh sodium hydroxide was added, and so on, so that a set of values for contact angle after 100 sec, induction time, and

Table 2 . . . Adsorption of Calcium by Quartz at Various pH Values

Calcium Concentrations	pH Value of Solution						
Mg per l	6.8	9.6	10.6	10.8	11.0	11.8	12.1
288	(0.002)	0.010	0.014		0.018	0.030	0.024
115				0.024			0.024

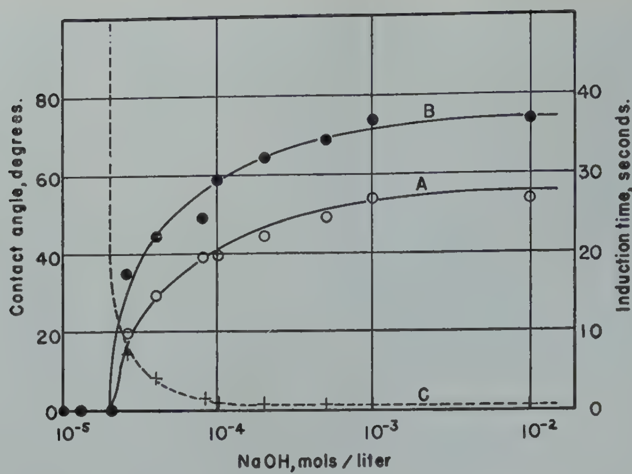

FIG 1—Contact angle and induction time as function of sodium hydroxide concentration. Sodium oleate, 20 mg per l; calcium ion, 100 mg per l. Curve *A*, initial contact angle; curve *B*, contact angle after 100 sec; curve *C*, induction time.

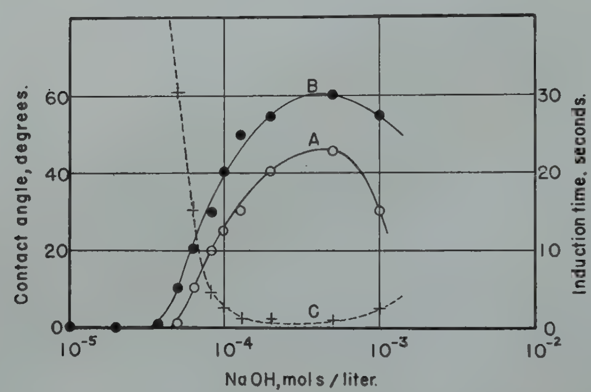

FIG 2—Contact angle and induction time as function of sodium hydroxide concentration. Sodium oleate, 20 mg per l; calcium ion, 1 mg per l. Curve *A*, *B*, and *C* same as in Fig 1.

initial contact angle was obtained for a wide range of sodium hydroxide concentrations.

In each succeeding test either the concentration of calcium chloride or sodium oleate was changed during the run, while the concentrations of the two remaining reagents were held constant. The influence of bubble age up to 100 sec was also observed, and of different reaction times before bubble application. A few tests were made in which the specimens were pretreated with a solution of calcium chloride and sodium hydroxide, and then transferred to the bubble cell which was filled with a solution of sodium oleate and sodium hydroxide.

RESULTS

When the quartz was pretreated in calcium solutions and then tested in the cell with no calcium present, a contact angle was obtained immediately after transfer, but it decreased rapidly with elapsed time, and after a few minutes no contact could be obtained. The calcium was probably desorbed under these conditions. When calcium was added to the cell together with the other reagents, the contact angle increased, and the induction time decreased in the first minutes of elapsed time. When tested after an elapsed time of 3 min or more, no observable change in the values was found. For bubble ages ranging from 1 to 100 sec, no changes in the observed values could be found.

Addition of small amounts of pine oil, up to 10 mg per liter, did not change the values of either contact angle or induction time. When a reaction time of 3 min or more was employed, tests in which the sequence of reagent addition was changed gave identical results.

The results of two tests with sodium hydroxide concentration as the variable are shown in Fig 1 and 2, of one test with changing calcium chloride in Fig 3, and of one test with sodium oleate concentration as the variable in Fig 4. The decrease in contact angle at high sodium hydroxide concentration shown in Fig 2, should be noted, and will be discussed later.

DISCUSSION

For the system of quartz in a dilute solution of calcium chloride, sodium hydroxide and sodium oleate at a pH ranging from 6 to 11, the following conclusions may be drawn:

1. Equilibrium is obtained within a few minutes, even for desorption of cal-

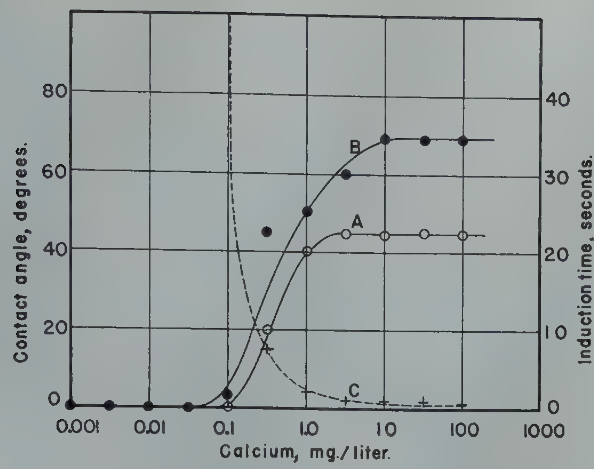

FIG 3—Contact angle and induction time as function of calcium ion concentration. Sodium oleate, 20 mg per l; sodium hydroxide, 40 mg per l. The pH changed from 10.8 to 9.8 during the test. Curve *A*, *B*, and *C* same as Fig 1.

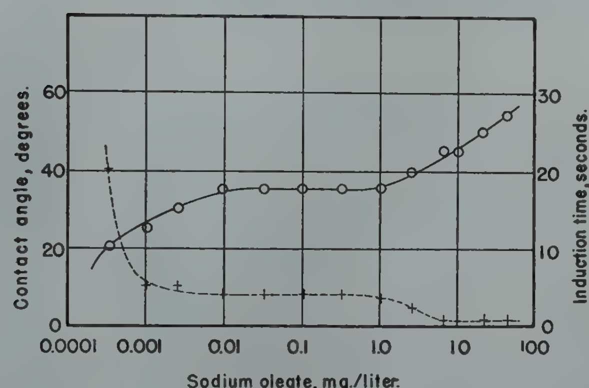

FIG 4—Contact angle and induction time as function of sodium oleate concentration. Calcium ion, 100 mg per l; sodium hydroxide, 80 mg per l; pH, 11.2. Upper curve, initial contact angle; lower curve, induction time.

cium oleate on the quartz surface.

2. Contact angle and induction time are roughly inversely proportional.

3. Under the correct conditions of pH and calcium concentration, extremely small amounts of sodium oleate, less than 0.001 mg per liter, may give contact.

The curves of Fig 1 to 4 show that in all cases a broad transition zone exists between no contact and full contact angle.

Bubble Pick-up Method

To investigate the behavior of crushed quartz particles compared with polished quartz in some of the preceding tests, a small amount of vein quartz crushed to minus 1 mm was added to the bubble cell. After each reagent addition an air bubble was pressed against the particles, and the quantity of particles that was picked up, that is, adhered to the bubble when it was lifted, was observed.

Such tests were made during the runs shown in Fig 1, 2, and 3, and the following general observations were noted: when the concentration of the added reagents was too small to give contact with polished quartz, no adherence (henceforth referred to as "pick-up") occurred. In the region where a slight contact angle was obtained, an attraction of the small particles to the bubble could be observed but no particles adhered to the bubble when it was lifted. When the reagent additions were sufficient to effect a contact angle of from 20 to 40°, with an induction time of from 10 to 5 sec, the smaller particles, about 0.1 mm diam, adhered to the bubble when it was raised, that is, a partial pick-up was observed. Finally when the conditions were such that the contact angle was from 30 to 60°, all the quartz particles could be picked up by the bubble, and the underside of the bubble could be completely coated with quartz particles. Induction times for the pick-up test were in all cases less than 1 sec.

The reagent concentrations required to give complete pick-up with the particles and a contact angle of at least 60°, with the polished quartz, and the maximum concentrations to give no pick-up and no contact, respectively, are given in Table 3.

Examination of Table 3 shows that even this crude pick-up method is more sensitive to changes in reagent concentration than the contact angle method. It was decided to investigate further the possibilities of this method, and the technique used and the results obtained are described below.

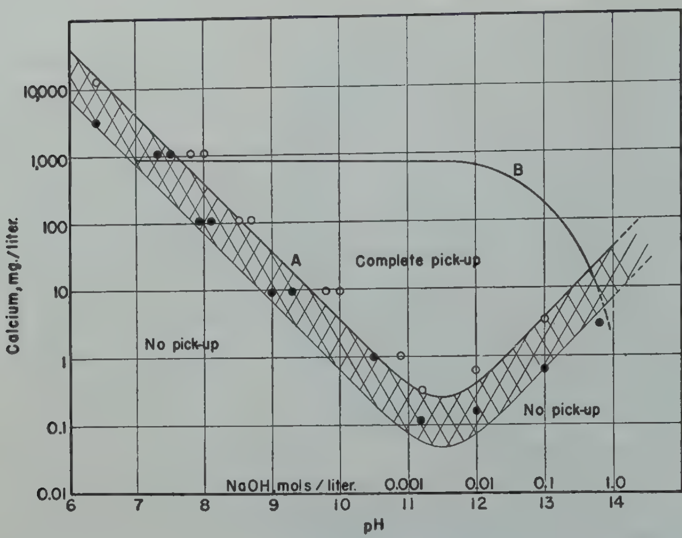

FIG 5—Pick-up for different concentrations of calcium ion and sodium hydroxide. Sodium oleate, 20 mg per l. Open circles, complete pick-up; filled circles, partial pick-up. Curve *A*, limit of complete pick-up; curve *B*, approximate limits of calcium hydroxide solubility after d'Anselme.[14]

Water-clear vein quartz from Montana was crushed dry though 1 mm, elutriated with distilled water, and the fraction plus 0.2 mm used for the tests. This fraction was leached overnight in $1N$ hydrochloric acid, washed, and leached another night with $1N$ sodium hydroxide. The material was then washed eight times with distilled water and stored under distilled water until used. Identical results were obtained when freshly leached samples and samples that had been stored for three weeks were used.

Various types of cells were constructed, and finally a simple test tube-shaped cell was adopted, consisting of a centrifuge vessel with a rubber stopper and a glass tube extending through the stopper to near the bottom. On the top of the tube was attached a piece of rubber hose closed at its upper end with a hose clamp, this hose acting as an air chamber for generating the bubbles. This design of cell can be improved, and details of a new type of cell will be given in a subsequent paper.

METHOD OF MAKING PICK-UP TESTS

The cell was filled with 30 ml of distilled water, and about 0.05 g of wet quartz particles was transferred to the cell. These particles then were tested for pick-up with the air bubble. In some cases one or two particles were picked up, but it was easy to distinguish this from the pick-up obtained under activating conditions, and during the tests such behavior was ignored. The next step was to add calcium chloride and sodium oleate in the desired quantities, and to test for pick-up again. For concentrations of calcium less than 5 g per liter, no pick-up was observed at a pH of 6.7. Then a small amount of sodium hydroxide was added, the cell was shaken and the pick-up was tested after a time lapse of 3 min. If there was any pick-up its degree was noted and the cell was then tapped several times with the finger. In some cases the particles adhered so loosely to the bubble that they became detached by this tapping, or even by the small agitation caused by the raising of the bubble in the tube. After noting the degree of pick-up after tapping, more sodium hydroxide was added, usually sufficient to double its concentration, and the pick-up test was repeated again. In this way each test run gave values for the pick-up over a wide range of sodium hydroxide concentrations.

Table 3 . . . Comparison of Pick-up and Contact Conditions

Test	Variable Reagent	No Pick-up	Complete Pick-up	No Contact	60° Contact
Fig 1	NaOH	0.00002 mol per l	0.00004 mol per l	0.00002 mol per l	0.0001 mol per l
Fig 2	NaOH	0.00005 mol per l	0.0001 mol per l	0.00005 mol per l	0.0006 mol per l
Fig 3	Ca^{++}	0.1 mg per l	1.0 mg per l	0.03 mg per l	3.0 mg per l

The concentrations at which pick-up commenced and at which it was complete were of special interest. The pH was measured at these concentrations by withdrawing about 5 ml of the solution and measuring the pH in a Coleman glass electrode pH meter. The values so obtained agreed well with colorimetric determinations, except for the ranges covered by thymolblue and bromthymolblue, which reacted in some way with the oleate ion. In some tests the concentration of calcium was varied, and the concentrations of sodium hydroxide and sodium oleate were kept constant.

RESULTS

Fig 5 summarizes the results of tests made with different concentrations of sodium hydroxide and calcium chloride. The filled points show the commencement of pick-up, the open circles complete pick-up. The values given are for pick-up after tapping the cell, as this method gave the most consistent results by eliminating the influence of slight vibrations which could not be controlled or evaluated. In most cases the difference between results with and without tapping were slight.

For the curve given the sodium oleate concentration was 20 mg per liter. Tests with 1 mg per liter gave practically the same results, but the transition from commencement of pick-up to complete pick-up was more gradual. Addition of sodium chloride had no influence on the results when the mol concentration of sodium ions present was less than from 400 to 800 times the mol concentration of calcium ions. Higher concentrations of sodium ions decreased the pick-up and when it exceeded 8000 times the calcium ion concentration pick-up was prevented almost completely.

The tests with sodium bicarbonate showed that when the concentrations and the pH are such that a precipitate of calcium carbonate is formed, there is no pick-up. Smaller concentrations of bicarbonate to some extent decrease the pick-up.

The tests with constant sodium hydroxide and variable calcium chloride concentration disclosed that for a pH between 6 and 9 the pH is very unstable, undoubtedly because of adsorption of carbon dioxide from the air admitted when the stopper was removed for reagent addition. The design of the new cell should obviate this difficulty. Care should be taken,

when a new cell is used, that activating ions released from the glass do not vitiate the results. This can be taken care of by using either Pyrex glassware, or if soft glass is used, by treating the cell with a solution of high pH until all available ions are leached out.

DISCUSSION

If it is accepted that the pick-up tests delineate those conditions under which quartz will be activated, then the following conclusions may be drawn:

1. Both hydrogen ions and sodium ions influence the activation of quartz with calcium ions. The higher the concentration of hydrogen ions and sodium ions, the more calcium ions are required to activate the quartz. From the results, the following approximate relationships were calculated:
   a. There is complete activation for quartz with calcium when: the mol concentration of Ca is greater than $[H^+] \times 10^6$. the mol concentration of Ca is greater than $[Na^+] \times 10^{-3}$.
   b. There is no activation of the quartz with calcium when: the mol concentration of Ca is less than $[H^+] \times 10^5$. the mol concentration of Ca is less than $[Na^+] \times 10^{-4}$.
2. When both sodium and hydrogen ions are present, then the combined effect of the ions must be taken into account, and we have the expression:

mol Ca concentration

$$= 10^6[H^+] + 10^{-3}[Na^+]$$

This expression gives the minimum amount of calcium necessary to give complete activation of quartz. Curve *A* of Fig 5 represents this expression, and it may be seen that the points determined by the pick-up tests are as close to this curve as may be expected from the accuracy involved. The shaded area on Fig 5 indicates when incomplete pick-up and activation occur. The deactivating effect of sodium ions also explains the decrease in contact angle test shown in Fig 2.

Summary and Conclusions

In 1937, Fahrenwald and Newton, in a most comprehensive paper,[2] showed that:

> . . . calcium ion, Ca^{++}, is a strong precipitant of quartz only in the presence of hydroxyls. Hydroxyl anions associate with and remove hydrogen cations, and calcium silicate is formed which is less soluble in alkaline than in acid solutions. Ca^{++} does not precipitate quartz in acid solution for reasons the reverse of those just stated—that is, ionization of silicic acid is suppressed and calcium silicate increases in solubility.

Elsewhere in the same paper they state: "In the absence of hydroxyl ions, H-ions resist tenaciously the attacks of calcium cations, and they give ground only to relatively high concentrations of cations." These generalizations, determined from settling rates, have been confirmed and quantified by the writers, using a modified captive bubble method.

In 1942, Gaudin and Rizo-Patrón,[11] as a result of certain experiments, concluded "that the barium ion is not abstracted by quartz from an acid circuit" and on theoretical grounds deduced that the number of loci of cationic preference on the surface of crushed quartz should amount to 7 micromols per square meter. The writers find that calcium behaves similarly to barium with respect to the first statement, and that the maximum adsorption of calcium on quartz determined experimentally amounts to 7.1 micromols per square meter.

The following picture of the adsorption of cations on the surface of quartz is proposed: all cations present in the solution tend to adsorb on the surface, and the preferential sequence is H^+, Ca^{++}, and Na^+. To displace the adsorbed hydrogen ion to any degree, the calcium concentration in the solution must be more than 10^5 times the hydrogen ion concentration, and the sodium ion concentration must be more than 10^9 times the hydrogen ion concentration. Sodium ions will commence to displace calcium ions at a concentration about 10^3 times the calcium ion concentration. As of these three ions, only adsorbed calcium ions will cause flotation with a soap collector, both hydrogen ions and sodium ions can act as depressants for calcium-activated quartz.

It should be noted that the minimum quantity of calcium necessary for activation occurs at a pH of 11.5 (Fig 5). Presumably this may be true only when sodium hydroxide is employed for pH regulation, for if some other alkali metal hydroxide were used it does not necessarily follow that the metal ion will displace calcium at the quartz surface. Actual flotation tests on quartz, using close to 75 mg Ca^{++} per liter of water, sodium hydroxide, with sodium abietate and mineral oil as collector have given 100 pct recovery at a pH of 11.5, with the recovery slowly approaching this value from the acid side, and sharply dropping once it is exceeded. The results of these tests will be given in a later paper.

Acknowledgments

The writers wish to express their appreciation to Mr. Norman F. Schulz, who made the surface determinations of the quartz used in the adsorption experiments, and to Dean Thomas L. Joseph for his active interest in the prosecution of this work.

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The Flotation of Quartz Using Calcium Ion as Activator

By STRATHMORE R. B. COOKE,* Member AIME

On the basis of experiments conducted on quartz using a bubble pick-up method, it was shown in an earlier paper[1] that this mineral will preferentially adsorb hydrogen, calcium, or sodium ions, depending on the relative concentrations of those ions in the solution in which the quartz is immersed. For quartz particles ranging in size from 0.2 to 1 mm, it was demonstrated that the concentration of calcium in solution (assumed present as ions) necessary to completely activate quartz for flotation is given by the expression:

$$Ca^{++} = [H^+] \times 10^6 + [Na^+] \times 10^{-3}$$

In this expression, ionic concentrations are in mols per liter. It was further shown that the pick-up method is apparently more sensitive to changes in reagent concentration than the standard captive-bubble method, and that induction times are apparently much reduced.

Since completing these earlier tests, a new type of cell has been constructed; this is shown in Fig 1 and 2. In Fig 1, *A* is a ground joint, *B* is a central tube reaching to within a few millimeters of the bottom of the cell, and *C* is a stopper which may be removed for reagent addition, and serves the further purpose of excluding carbon dioxide from the air during the test. The entire cell is constructed of Pyrex glass, and does not give the trouble experienced with the earlier cell, in which activating ions were released from the glass at high pH values. The only critical factor in the construction of the cell is the clearance between the central tube and the bottom of the cell. This clearance should be sufficiently small that the bubble can be pressed directly on the mineral grains lying on the bottom of the cell.

In the pick-up tests to be described in this paper, the reagents used were all of C. P. grade, except the sodium oleate, which was Merck's "neutral powder." The quartz employed was water-clear vein quartz, sized on screens, cleaned with both hydrochloric acid and sodium hydroxide, and given a thorough final washing with distilled water. Experimental procedures were the same as described in the earlier paper.

EFFECT OF SIZE OF QUARTZ ON PICK-UP

To ascertain the effect of particle size on the adsorption of calcium ions, the quartz was sized from minus 14 plus 20 mesh through the intervening screen sizes to minus 270 mesh plus 400 mesh. Each size was thoroughly cleaned, and then tested in the cell at different calcium chloride and sodium hydroxide concentrations, and at a constant sodium oleate concentration of 20 mg per liter.

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TP 2607 B. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Sept. 30, 1949. Manuscript received Dec. 3, 1948.

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[1] References are at the end of the paper.

All particles, within the size range given, exhibited complete pick-up within the curve expressed by the equation above. This presumably means, when the conditions imposed by the equation are satisfied, that this maximum is independent of particle size.

However, it was found that the range through which *partial* particle pick-up occurred progressively broadened as particle size decreased. This is shown in Fig 3, in which curves *B*, *C*, and *D* show the limits at which pick-up just commences (as the pH is increased) for particles of minus 14 plus 28 mesh, minus 65 plus 100 mesh, and minus 270 plus 400 mesh size, respectively. These results indicate that for satisfactory activation, at any given pH, a lower calcium ion concentration is required for fine particles than for coarse particles.

EFFECT OF HIGH ALKALINITY ON PICK-UP

At calcium concentrations of between 1 and 10 mg per liter, and at high alkalinities, it was noticed that pick-up ceased as soon as calcium hydroxide commenced to precipitate. This effect was investigated at other calcium concentrations, with the same results. Solutions of calcium chloride, containing 10^5, 10^4, 10^3, and 10^2 mg of calcium per liter were made alkaline with sodium hydroxide until calcium hydroxide just started to precipitate, according to the following equation:

$$CaCl_2 + NaOH \rightarrow Ca(OH)_2 + 2NaCl$$

The beginning of precipitation was taken as that point at which either a faint opalescence appeared in the solution, or a Tyndall cone became ap-

parent. Referring to curve A in Fig 3, the triangular point at a pH of 7.0 and a calcium concentration of $10^{5.47}$ mg per liter represents the solubility of calcium chloride hexahydrate (the equilibrium form at 25°C, the temperature of the experiment). The remaining triangular points give the pH at which calcium hydroxide started to precipitate, for each designated concentration of calcium. A smooth curve can be drawn through these points. It will be noted that the transition between complete pick-up and partial pick-up agrees rather well with this curve, except for the points at 10^5 mg of calcium per liter, where the heavy calcium hydroxide precipitate obscured the pick-up tests. It seems probable that depression of quartz along this curve is due to competition between hydroxyl and oleate ions for the adsorbed calcium ions on the quartz. When hydroxyl replaces oleate ion the mineral becomes hydrophilic and is depressed.

With hydroxyl, oleate, and calcium ions present, there is the possibility of calcium hydroxide and calcium oleate forming. According to the law of mass action:

$$\frac{[Ca^{++}][OH']^2}{[Ca(OH)_2]} = K_h \qquad [1]$$

and:

$$\frac{[Ca^{++}][C_{18}H_{33}O_2']^2}{[Ca(C_{18}H_{33}O_2)]} = K_o \qquad [2]$$

Dividing Eq 1 by 2:

$$\frac{[Ca(C_{18}H_{33}O_2)][OH']^2}{[Ca(OH)_2][C_{18}H_{33}O_2']^2} = \frac{K_h}{K_o} = K \qquad [3]$$

Eq 3 states that at constant oleate ion concentration, increase in pH requires either a decrease in calcium oleate concentration or an increase in calcium hydroxide concentration, and this is in conformity with the depression of quartz along curve A in Fig 3. The constant K is independent of the calcium ion concentration, however, so that at constant pH and oleate ion concentration there should be no such closure of curve A with curve E as is shown at the low pH end of Fig 3. Either the above reasoning or the experimental results are open to question.

FLOTATION TESTS

A series of flotation tests was made to ascertain the correlation between conventional flotation and the pick-up results. Because of the unavailability at the time of massive quartz, the flotation tests were made on Ottawa sand.

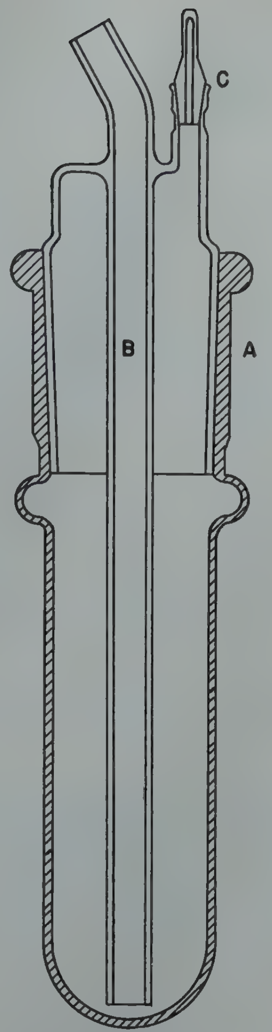

FIG 1—Cross section of cell used in pick-up tests.

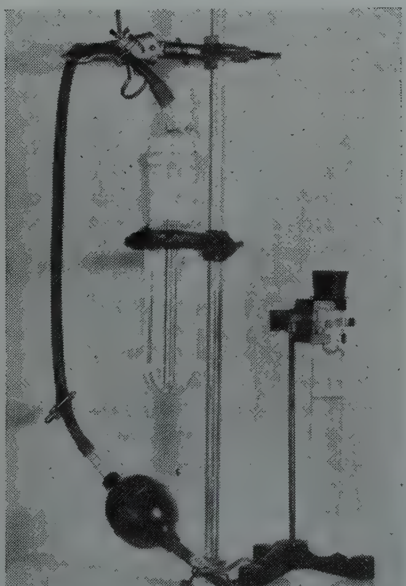

FIG 2—Pick-up cell.

The 250 g samples of sand, with a size range of minus 28 mesh, plus 48 mesh, were boiled for 10 min with 100 ml of concentrated hydrochloric acid and 50 ml of water to leach out iron and calcium. The leached sand then was washed ten times by decantation, filtered, and washed with distilled water until the washings gave no test for chloride ion with silver nitrate. Each lot was prepared for flotation by grinding in a pebble mill for 15 min at a fixed dilution with distilled water, filtering through a Buechner funnel, washing with a series of applications of boiling distilled water, and then transferring to the flotation cell. The pH of the distilled water used in grinding was 5.6, and that of the water filtered from the grind was uniformly 6.5.

The flotation cell used was identical with that described by Clemmer and Clemmons[2] except that it was fabricated of stainless steel rather than of bakelite. To maintain constant pH in the cell during a run, the froth continuously was removed, filtered, and the filtrate returned to the cell. Under these conditions, the pH at commencement and conclusion of each test differed very little except when the initial pH was 8.0 or less. Variation in pH near neutrality is attributed to carbon dioxide taken from the air. Between tests, the cell was given a number of thorough washes, first with a hot solution of sodium carbonate, and then with boiling distilled water. This treatment was always found necessary, because stainless steel seems to tenaciously retain traces of soap. Flotation procedure consisted in diluting the pulp to cell volume with distilled water, agitating, and then measuring the pH on a Coleman meter.

This was always close to 6.2, an increase in pH due probably to the effect to which Gaudin has called attention.[4] Sodium hydroxide solution was then added in sufficient quantity to give the desired pH, the cell was operated as a conditioner for 5 min, and a check pH was taken. Calcium chloride solution then was added in the required amount, the pulp conditioned for another 5 min, the pH once more measured, and the collectors, sodium abietate or sodium oleate and mineral oil, then added. After another 2 min of conditioning, the air was admitted, and flotation allowed to proceed for 8 min. No departure from these conditions occurred in the tests, the results of which will be given. All floatable mineral was removed within the 8 min. The sodium abietate was not chemically pure, and was prepared from an industrial product, Tallex.

Results of the tests are shown in Fig 4. Curve A shows the recovery percent-

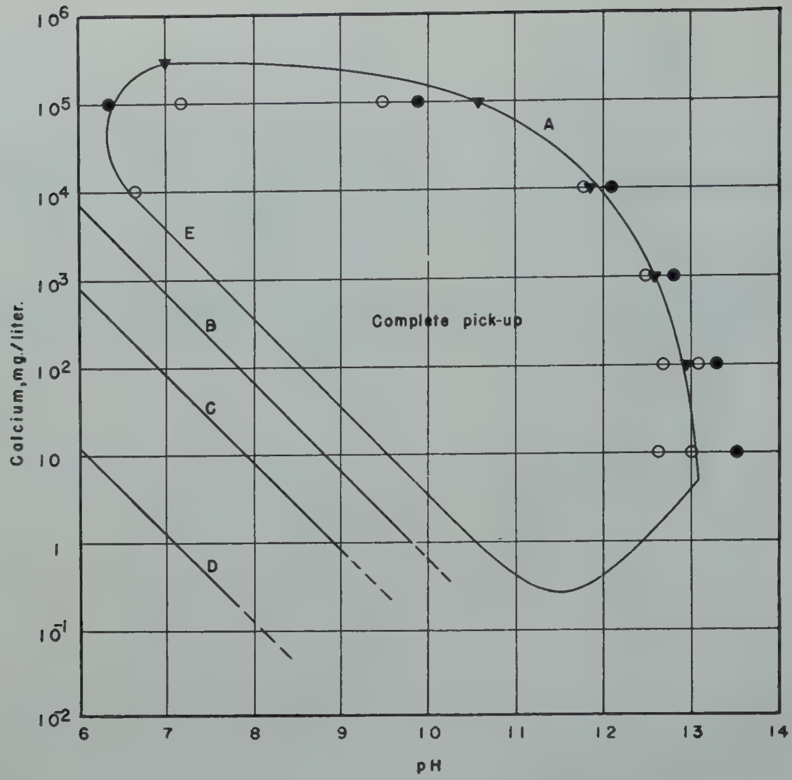

FIG 3—Pick-up of quartz for various concentrations of calcium ion at pH values above 6.0. Sodium oleate concentration, 20 mg per liter. Open circles indicate complete pick-up, filled circles no pick-up. Curve *A*, precipitation limits of calcium hydroxide (triangular points); curves *B*, *C*, and *D*, limits at which pick-up just commences for $-14+28$, $-65+100$, and $-270+400$ mesh quartz; curve *E*, limit of complete particle attachment for all sizes of quartz tested.

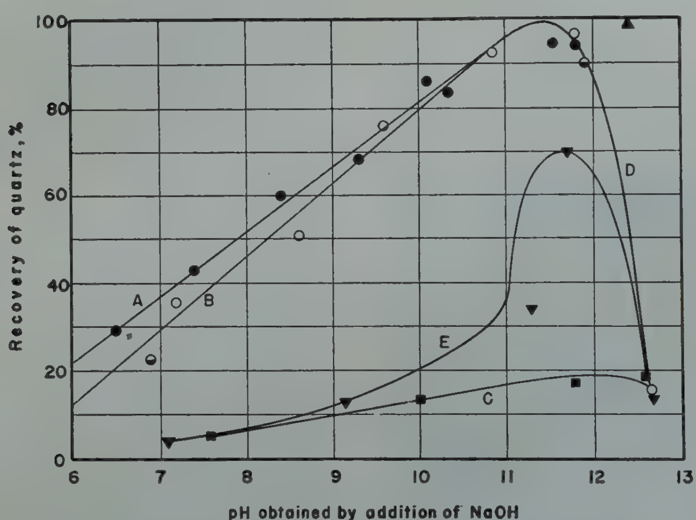

FIG 4—Per cent recoveries, of clean quartz floated with calcium ion as activator at various pH values. Curve *A*, 1.0 lb $CaCl_2$, 1.5 and 3.0 lb sodium abietate, and 1.36 lb mineral oil, per ton of quartz: curve *B*; 0.5 lb $CaCl_2$, 1.5 lb sodium abietate, and 1.36 lb, mineral oil per ton of quartz: curve *C*; no calcium, 1.5 lb sodium abietate, 1.36 lb mineral oil, per ton of quartz: curve *D*; depression of calcium-activated quartz by sodium ion: curve *E*; 1.0 lb $CaCl_2$, 1.5 lb sodium oleate, and 1.36 lb mineral oil per ton of quartz.

age of quartz, using a constant quantity of calcium chloride (1.0 lb $CaCl_2$ per ton of ore, or at the dilution employed, 75 mg of calcium ion per liter of water), and both 3.0 and 1.5 lb of sodium abietate per ton of ore. Curve *B* represents the recoveries using 0.5 lb of calcium chloride and 1.5 lb of sodium abietate per ton. In all cases 1.36 lb of heavy white mineral oil was used. Curve *C* cannot be explained by calcium activation, for the quartz was treated by acid. It is possible that ferric ion is responsible, however.

Curve *E* represents the flotation of quartz with 1.0 lb of calcium chloride and 1.5 lb of sodium oleate per ton of ore and under conditions identical with those using sodium abietate as collector. No attempt was made to ascertain optimum quantity as collector. It will be noted that maximum recovery of quartz occurs at a pH of 11.5 for both curves *A* and *B* and close to this, considering the limited number of points, for curve *E*. The depressing effect of sodium ion is well shown by curve *D*, and the high alkaline part of curve *E*. It should be remembered, however, that for each change of one unit in pH, there is required a tenfold change in sodium hydroxide concentration, assuming 100 pct ionization, so that large quantities of alkali are required to attain high pH values. That depression is not due to hydroxyl ion is demonstrated by the isolated point at a pH of 12.4. In this test the quartz was floated with 1.5 and 1.36 lb per ton, respectively, of sodium abietate and mineral oil. No sodium hydroxide was used, the quartz being floated from a saturated solution of calcium hydroxide. The quartz recovery was 99.6 pct, and the calcium concentration initially in the liquid was 840 mg per liter. This would correspond to a point almost exactly on curve *A* of Fig 3.

SIZE OF QUARTZ PARTICLES FLOATED

It was noticed, at pH values other than optimum, that the froth was apparently normal, but that the particle size floated decreased rapidly with departure from a pH of 11.5. This was due not to fine particles being carried over in the interbubble liquid, but to a real decrease in the size of the particles being floated. Doubtless every flotation operator has noticed similar behavior when starvation quantities of reagent have been used.

To investigate this, sizing analyses of concentrates and tailings were made at recoveries of 21.9 and 63.5, respectively. Data from the first-mentioned test are given in Table 1, together with per cent recoveries for each size for the second test. Fig 5 shows the per cent recovery of each size plotted against size for both tests. It is obvious that at the lower recovery only fine material was floated. In both tests the recovery

in the finest product was slightly less than in the immediately preceding size fraction, due no doubt to failure of flotation in the minus 10 micron range, demonstrated elsewhere. For the test made at a recovery of 63.5 pct a sizing analysis was unfortunately not made below 400 mesh.

The results indicate that, at any given pH, less calcium ion is required in solution to activate small particles than is required to activate large particles. This confirms the results obtained with the pick-up method.

With the dilution used in the flotation cell, 1.0 and 0.5 lb of calcium chloride correspond to 75 and 37.5 mg of calcium ion per liter. Reference to Fig 3 shows that at these concentrations, *all* the quartz should have floated at pH values of approximately 8.6 and 9.0, respectively. It is obvious from Fig 4 that this did not happen. However, in the pick-up tests, the small surface of the quartz grains could have no perceptible effect on changing the calcium ion concentration in the solution. In the case of the flotation tests, however, the situation is different. Here the total surface of the quartz is large compared with the number of calcium ions available, and as the quartz adsorbs calcium ions until equilibrium is reached, calcium ion concentration in the solution will drop. Thus the curve of Fig 3 may not be directly used to predict flotation results.

From the sizing analysis given in Table 1, and using the correction factors of Gross and Zimmerley[3] and of Gaudin and Rizo-Patrón,[4] the approximate surface area of the plus 13 micron flotation feed is calculated to be 170,000 sq cm. It is difficult to assess the average size of the minus 13 micron fraction. Assuming it to be 1 micron, then with a shape factor of 1.5, the approximate surface area of the 16.63 g of material would be 564,000 sq cm, or a total of 734,000 sq cm for the 250 g of flotation feed. It is more probable that this is an underestimate of total surface rather than an overestimate.

Assuming that quartz, under *optimum conditions*, adsorbs 7.0 micromols of calcium ion per square meter, then the 250 g of flotation feed could adsorb:

$$\frac{7.34 \times 10^5 \times 7.0 \times 40 \times 10^{-6} \times 10^3}{10^4}$$

or 21 mg of calcium ion.

In a qualitative way, the rough calculation given above serves to explain why the direct flotation tests apparently do not coincide with the data of Fig 3. Another possible source of discordance is the difference between the swirling action of the flotation cell and the quiescent conditions within the pick-up cell, which would be expected to adversely affect flotation of the larger particles in the flotation cell.

Table 1 . . . Per Cent Recoveries of Quartz at Various Sizes

Size	Concentrate, Weight, g	Composite Head, Weight, g	Per Cent Recovery in Concentrate (a)	Per Cent Recovery in Concentrate (b)
− 35 + 48 mesh	0.00	0.26	0.0	0.0
− 48 + 65 mesh	0.00	2.43	0.0	3.8
− 65 + 100 mesh	0.23	21.90	1.1	7.5
−100 + 150 mesh	1.41	59.55	2.4	34.6
−150 + 200 mesh	2.74	46.25	5.9	59.4
− 74 + 52 microns	9.47	46.18	19.8	75.2
− 52 + 37 microns	9.09	23.40	38.8	91.6
− 37 + 26 microns	8.70	15.39	56.5	84.7
− 26 + 18.5 microns	7.52	10.91	68.8	
− 18.5 + 13 microns	4.92	7.13	69.0	
− 13 microns	11.41	16.63	68.8	

a. Flotation test at pH 6.5, recovery 21.9 pct
b. Flotation test at pH 9.3, recovery 63.5 pct

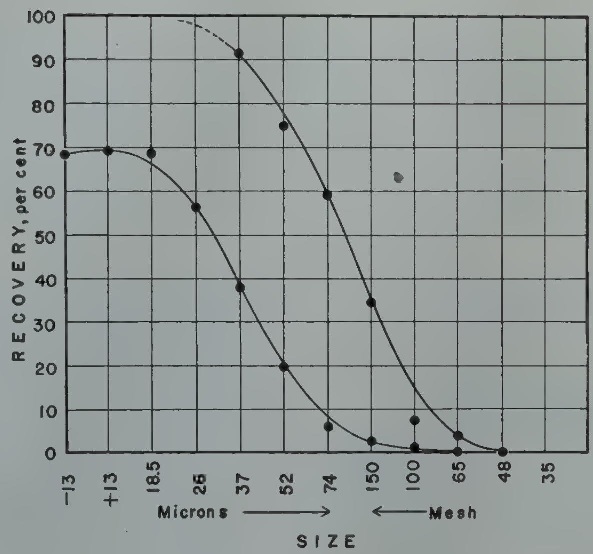

FIG 5—Size fractions in quartz concentrates prepared at pH 6.5 (recovery 21.9 pct, lower curve) and pH 9.3 (recovery 63.5 pct, upper curve), plotted against per cent recoveries for each size. At 100 pct recovery of quartz, the curve would be a straight line through the 100 pct point, parallel to the abscissa.

particles can adsorb calcium ions under conditions which prohibit large particles from doing the same. Because of the preliminary nature of the investigation, no attempt is made to discuss the fundamental reasons for the phenomena observed.

Summary

In a general way, the activation and flotation of quartz, using calcium ion as the activator, may be explained on the basis of exchange of hydrogen ions on the quartz and calcium ions in solution. At high sodium ion concentrations, sodium will depress calcium-activated quartz by removing the calcium. At high calcium ion concentrations, hydroxyl ions can replace collector ions and depress the quartz. Evidence is presented that fine quartz

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Some Economic Aspects of Perlite

By C. R. KING,* Member AIME

Introduction

Most of the acid volcanic glasses such as obsidian, perlite, pitchstone, pumice, and pumicite (volcanic ash) are susceptible to some expansion if suddenly subjected to a suitably high temperature in a properly designed furnace, and subsequently suddenly cooled. Expansion results from the action of chemically combined or dissolved water expanding as a vapor within the softened heated rock particles. This expansion forms countless minute bubbles within the pasty mass of the natural glass particles. If the particles are cooled before these bubbles can escape, the resulting products are cellular, more or less spherulitized glass froths or hollow glass spheres of many types. Chemically, they are essentially aluminum silicates containing about 70 to 75 pct silica, 10 to 16 pct alumina, and 2 to 7 pct sodium plus potassium oxides. They are relatively resistant to attack by acids or alkalies, are resistant to heat up to over 1000°F, and are completely inert to insect or fungus attack.

These products are of great potential value in many branches of industry as insulation materials, aggregates, fillers, abrasives, and additives. By proper selection of the perlite rock type and control of processing conditions, expanded perlite of a bulk density varying from about 2 to 20 lb per cu ft can be made. The physical properties of the particles comprising a product of a given bulk density also may be varied within wide limits of porosity, water absorption, and crushing strength.

In general, crushing strength of expanded perlite increases in direct relation to increase in bulk density, but insulating value varies in inverse ratio to bulk density. The selection of a particular grade of expanded perlite for a given use as an aggregate is therefore always a compromise between insulating value, lightness, and friability. If porosity and water absorption are factors affecting the use of the material, factors other than bulk density are important. Some types of perlite rock tend to decrepitate excessively during expansion, resulting in a highly porous or even hygroscopic product of high friability relative to the bulk density of material produced. In general, treatment of any good perlite rock below the temperature necessary for maximum expansion, or insufficient time in the hot zone of the furnace for effective heat penetration to the core of the rock particles will result in some decrepitation and high porosity in the expanded product. On the other hand, exposure of the material to the temperature required for optimum expansion for a period of time slightly longer than that necessary for proper heat absorption will result in a slightly heavier product of extremely low porosity, i. e., a glazed surface. It is beyond the scope of this paper to describe in detail the rock types and processing methods used to produce the many types of expanded perlite in present use, or that may be produced to fill the present and future needs of industry. A few phases of the economics involved in processing and selling products made from this abundant and cheap mineral raw material, plentifully distributed in large and relatively accessible deposits in most western states and in other areas within economic shipping distance of large centers of consumption, will be treated with particular reference to the Los Angeles area.

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Relation of Bulk Density to Use and Cost of Production

Perlite differs from most other lightweight aggregates and fillers in that the raw material (perlite rock) may be quarried much as rock and sand is mined, and then shipped into consuming areas as crude rock in open gondola cars at correspondingly low freight rates and handling costs. At processing plants in or near large consuming areas the rock is converted into relatively fragile, very bulky products which in most cases are used within a short haul of the processing plant or enter into a manufactured product made at the site of the perlite processing plant.

Table 1 . . . Conversion Table, Cost per Ton to Cost in Cents per Cubic Foot of Expanded Perlite

Bulk Density, Lb per Cu Ft	Cu Ft per Ton	Cost, Cents per Cu Ft in Bulk at Plant, Case: (1) $15	(2) $18	(3) $25	Cost, Cents per Cu Ft in Bags at Plant,[a] Case: (1) $15	(2) $18	(3) $25	Grade of Material and Field of Use
2	1,000	1.5	1.8	2.5	7.5	7.8	8.5	
4	500	3.0	3.6	5.0	9.0	9.6	11.0	Grade 1, special purpose material
6	333	4.5	5.4	7.5	10.5	11.4	13.5	
8	250	6.0	7.2	10.0	12.0	13.2	16.0	Grade 2, plaster aggregate
10	200	7.5	9.0	12.5	13.5	15.0	18.5	
12	167	9.0	10.8	15.0	15.0	16.8	21.0	
14	143	10.5	12.6	17.5	16.5	18.6	23.5	
16	125	12.0	14.4	20.0	18.0	20.4	26.0	Grade 3, special purpose concrete aggregate, etc.
18	111	13.5	16.1	22.5	19.5	22.1	28.5	
20	100	15.0	18.0	25.0	21.0	24.0	31.0	

[a] Note: bagging cost has been allowed at 6 cents per cubic foot, which includes cost of bags, amortization on warehouse and equipment, labor and handling costs.

The possible fields of use for expanded perlite and the competition within those fields from other materials divide the various types of expanded perlite into three general categories which for convenience may be based upon the bulk density of the type of material suitable for use in those fields.

Grade 1 material falls within a bulk density range of 2 to 6 or 7 lb per cu ft. It is suitable for uses where very high insulating value against heat or cold coupled with extremely light weight are important, and relatively great friability is not an objection or is an asset (as in uses involving fine particle size abrasives or fillers: pulverizing is a simple matter).

Grade 2 material falls within a bulk density range of 7 to 10 or 12 lb per cu ft. It is suitable for uses where high insulating value and relatively light weight are desirable, but where use requires sufficient particle strength to withstand mechanical mixing with a stiff binder and hand or mechanical application to a mold or surface. Hardwall plaster aggregate is at present the largest single field for use for this grade of expanded perlite.

Grade 3 material includes all expanded perlite of a bulk density over about 10 to 12 lb per cu ft. A type of material within this range would be selected for any use where light weight and high insulating value are important, but where the aggregate is subjected to relatively rough treatment in mixing and application and where relatively high compressive strength in the finished product is imperative.

The question might arise as to the reasons for arbitrarily assigning definite weight per cubic foot ranges to the three general grades of expanded perlite noted; or the reasons for picking three categories rather than more or less, since the change in physical properties corresponding to variation in bulk density is entirely gradational. The reasons for thus assigning certain ranges in bulk density to definite fields of use are largely economic. On a volume basis, the cost of producing expanded perlite increases in direct proportion to increase in bulk density. Also, as bulk density (and compressive strength of the material) increases, the competition from other cheap lightweight aggregates is keener but the potential volume of the market increases rapidly. These facts make it convenient to assign the ranges in bulk density to the three general categories as outlined.

Grade 1 material can sell at a relatively high unit price within a present comparatively limited volume market because of the unique and valuable qualities of the products. Grade 2, (plaster aggregate, etc.) must sell in competition with plaster sand in price, but the potential volume of the market is enormous. Grade 3 material may sell in a limited market at relatively high unit price for special purpose use; or may enter the large volume market for lightweight concrete aggregate in areas where it can sell on a competitive basis with other materials. These limitations are illustrated in the following estimates of production cost and possible producers selling price structure in the Los Angeles area.

Estimated Possible Production Costs

The estimates in Table 1 are based on the assumption that crude perlite rock of suitable type is shipped into the Los Angeles area from the closest known deposits and processed on a scale of 50 tons per day, or 15,000 tons per year. The estimates include cost of raw material, freight charges, amortization of capital outlay over ten years, all direct costs, items of 10 pct weight loss as dust, etc., process royalty, taxes, and fixed overhead. The estimates are made on the basis of cost f.o.b. plant, on a weight (cost per ton) basis. Cost upon a volume basis may be had from the conversion table for various bulk densities. Three estimates have been made: (1) cost, with raw material cost at a minimum, for a plant integrated into a factory using the expanded perlite as an ingredient in a finished saleable article or product; (2) cost, with raw material cost at a minimum, for a plant expanding perlite for sale on the open local market; and (3) cost, with raw material cost somewhat higher and processing cost higher than the other two cases, for a plant using a special type of perlite rock for conversion into special purpose expanded perlite.

Case 1: Estimated cost per ton in bulk at plant, $15.00. This is probably a minimum cost for the Los Angeles area at present.

Case 2: Estimated cost per ton in bulk at plant, $18.00. This is probably an average cost in this area, assuming an efficient operation and process.

Case 3: Estimated cost per ton in bulk at plant, $25.00. This is probably the highest cost that may be permitted in this area and still compete in most fields of use.

Possible Future Selling Price Structure

Grade 1 expanded perlite (2 to 6 lb per cu ft bulk density): The estimates of production cost of low bulk density types of expanded perlite shown in the foregoing do not include further processing such as grinding, sizing of the material after expansion, etc. Most uses for these types of perlite fall into two categories: relatively large volume use as loose fill or preformed insulating materials such as insulating board and shapes, refrigeration and heat insulation, etc.; or relatively small volume use (small volume in the sense of each particular product using these types of perlite as an ingredient) as fillers, abrasives, filter aids, additives, etc. These last types of use usually require some sort of processing in addition to simple expansion, and might include sizing of the expanded material, grinding to fine particle size, various coatings on the particles or other special processing. In practically all cases involving the use of grade 1 expanded perlite types, marked economies are indicated if the perlite processing plant is inte-

grated into the manufacturing plant fabricating the finished article or material using these types of perlite as a major or minor ingredient. It would be difficult to predict a price structure for these types of perlite if sold by a producer on the open market. Competition would in many cases be a minor factor, and the individual small volume market as well as the unique properties of these types of perlite would in many cases result in a relatively high unit price.

Grade 2 expanded perlite (bulk density from 7 to 10 lb per cu ft): In discussing this type of perlite, the present potentially enormous market as plaster aggregate overshadows other possible uses of medium bulk density expanded perlite. From Table 1, it is apparent that packaging cost is a major item in the total cost of the low and medium bulk density material; becoming progressively less as bulk density increases. Packaging cost is particularly important in the case of plaster aggregate, which must always be moved from the producing plant to the job or building site in one form or another. So far, the only attempt to reduce packaging cost in the case of expanded perlite plaster aggregate has been the preparation of premix plaster, in which the same container carries both binder and aggregate. It is probable that in the Los Angeles area some form of returnable container would reduce packaging cost appreciably in the case of small jobs, and some form of tank truck using light detachable bodies which may be left at the job might make considerable saving in packaging cost in the case of larger jobs. If such tanks or other types of large volume containers are used, some means of easily and quickly measuring batch volumes of perlite aggregate would have to be integral with the container.

It is probable that building code and other standards will be set up in the near future governing the specifications of expanded perlite used as plaster aggregate. Among these specifications will be one governing the minimum bulk density material that may be so used. Experience so far has demonstrated that the minimum bulk density of perlite plaster aggregate at which yield is reasonably high (breakdown of the aggregate particles in mixing and application is relatively slight) is about 7 to 8 lb per cu ft. Since overall costs on a volume basis decrease in direct proportion to decrease in bulk density, the conclusion is that the bulk density of expanded perlite plaster aggregate will stabilize at about 8 lb per cu ft. This material can be produced in the Los Angeles area at a cost somewhere between 6 and 10 cents per cubic foot in bulk at plant; or between 12 and 16 cents per cubic foot in bags, f.o.b. plant. This would imply a producers selling price range between 90 cents and $1.30 per four cubic foot bag, after allowing customary producers gross markup.

In view of the appreciable weight and labor saving possible when using perlite plaster aggregate of the right type, the foregoing price range should make perlite competitive with plaster sand in this area, even if the many other advantages of perlite plaster aggregate such as high insulating value, are discounted. There is no question as to the competitive status of perlite plaster aggregate in other areas which do not have the abundant and cheap local supplies of high grade plaster sand found in the Los Angeles area.

Grade 3 expanded perlite (bulk density over 10 lb per cu ft): For most uses as structural nonbearing concrete or preformed masonry concrete aggregate, expanded perlite should cost not more than 15 to 20 cents per cubic foot ($4.00 to $6.00 per yard) in order to compete with other light aggregates on a price basis in the Los Angeles area. Reference to Table 1 shows that while these costs should be met in this area, there is little room for the customary producers markup. It seems logical, therefore, that most expanded perlite used as concrete aggregate in prefabricated shapes and structural nonbearing concrete should be processed in perlite plants at the plants manufacturing the prefabricated shapes or readymix light-aggregate concrete. The obvious advantages resulting from bulk handling of the material, waste heat utilization in other stages in the fabrication of the products, and inclusion of the cost of the aggregate in the total cost of a finished product would seem to force such a setup.

In the foregoing, the advantages of integrating perlite expansion plants into those plants fabricating products using expanded perlite in one form or another have been stressed. Most manufacturers who might use expanded perlite in their products, but have neither the technical knowledge or process rights and experience, would, while realizing the possible economies, question the practicality of integrating a perlite processing plant into their operations. To date this has been a sound objection. The perlite industry, however, is rapidly approaching the stage where a manufacturer of, say, preformed insulating shapes, can make arrangements with a firm specializing in the processing of perlite whereby an adequate perlite expanding plant may be built so integrated with the manufacturing plant that all materials are handled in bulk, waste heat is utilized, and other obvious economies are realized. Payment of a nominal royalty will ensure technical supervision and training of operating personnel, guarantee of desired capacity, uniform specification of the product, and assured supply of suitable crude perlite rock. The overall cost of expanded perlite to the manufacturer under such an arrangement would be in line with the cost estimated in this paper. Such plants may be operated economically at capacities ranging from 100 lb per hr up to many thousands of pounds per hour, with no difference in the quality of the expanded perlite produced.

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The Mining, Milling, and Processing of Perlite

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Background

With the postwar emergency for new housing and for new industrial buildings, much research has been done on lightweight aggregates for use in concrete and plaster. The trend toward lighter weight aggregates to relieve the dead load on the higher, steel framework buildings started toward the end of the last century.

Progressively the requirements became more stringent, crushed bank slag, weighing 80 lb per cu ft as compared to 100 lb for crushed rock, was supplemented in some sections by foamed slags and expanded shales weighing from 40 to 60 lb per cu ft. In other sections cinders were used weighing from 40 to 50 lb, including sand. Sections favored with more recent vulcanism turned to pumice weighing from 30 to 60 lb, whereas other localities made use of diatomite, from 28 to 40 lb. Exfoliated vermiculite weighing from only 6 to 10 lb per cu ft came into use.

As the search for lighter weights in aggregates continued so did the search for other desirable properties such as nailability, ease of cutting or channeling, and good insulation to both sound and heat. This search eventually led to a widespread interest in the expansion of perlite, or similar volcanic glasses, with the result that in 1946, the Office of the Housing Expediter[1] listed eleven firms reporting the operation of processing plants either on pilot or commercial scale producing aggregates weighing from 2 to 16 lb per cu ft.

Geology

Perlite is one of the large family of rocks originating from a granitic or acidic magma and generally has a "rhyolitic composition with a marked perlitic structure."[2]

The name "perlite" is a derivation from "perlstein" originally given "to certain glassy rocks (hyaloliparites, hyalo-rhyolites) with numerous concentric cracks, from the fancied resemblance of broken out fragments to pearls."[3] The cause of the perlitic texture is somewhat controversial. Johannsen attributes the texture to strain set up in the glass by cooling and describes the finding of peculiar, glassy balls, called marekanite, more or less rounded and showing concave indentations, which may be left when perlite is broken. Frank Rutley, on the other hand, in 1881, while working on vitreous rocks from Montana, observed that there was incipient or partially started crystallization with segregation of water toward the uncrystallized portion in all specimens of perlite examined. Further he observed evidence of strain around focii of crystallization to which he ascribed the development of the spherical cracking which gives the rock its perlitic texture.[4]

Widespread over the western and southwestern United States where Tertiary vulcanism is exposed, perlite commonly is associated with andesitic, basaltic, and rhyolitic lavas with interbedded tuffs and ash of Eocene age although some deposits are associated with rocks assigned to the Pliocene age.[5,6] In nearly all cases the perlite overlies water laid tuff beds or agglomerates or breccias of explosive nature, an outstanding exception being the occurrence of perlite as the chilled selvage of andesite intrusions.[7,8]

Table 1 . . . Perlites of Steens Mountains[5]

SiO_2	67.05	68.66
Al_2O_3	14.91	14.44
FeO	1.48	1.28
Fe_2O_3	0.92	0.80
MgO	0.65	0.18
CaO	2.44	1.96
Na_2O	4.15	3.86
K_2O	3.04	3.28
H_2O (+105°C)	4.35	4.80
H_2O (at 105°C)	0.50	0.40
CO_2	none	none
TiO_2	0.34	0.25
P_2O_5	0.12	trace
S	trace	none
MgO_2	trace	none
Totals	99.95	99.91

Analyses by W. H. and F. Herdsman.

Table 2 . . . Perlites of Mutton Mountains[10]

SiO_2	73.79	73.28
Al_2O_3	12.40	12.55
Fe_2O_3	0.52	0.58
FeO	0.62	0.63
MgO	0.11	0.08
CaO	0.80	0.80
Na_2O	3.16	2.97
K_2O	4.84	5.00
H_2O(+105°C)	3.24	3.60
H_2O(−105°C)	0.25	0.19
TiO_2	0.09	0.09
P_2O_5	0.01	0.01
MnO	0.02	0.02
Totals	99.85	99.80

Analyses by James Kerr, University of Minnesota, Oct. 23, 1946.

Available chemical analyses of perlites from the western and southwestern United States indicate that they are varieties of leucorhyolites rather than of normal rhyolites, leucorhyolites having less than 5 pct of dark minerals, normal rhyolites having more than 5 pct. This conforms with Johannsen's observation that leucorhyolites are the prevailing kind in North America.[9] Tables 1 and 2 give

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[1] References are at the end of the paper.

analyses of samples from the Steens Mountains and from the Lady Frances mine in the Mutton Mountains, Oregon.

Comparison of these perlite analyses with analyses of obsidian and pitchstone shows very little difference in composition, the principal difference being in percentages of combined water. Table 3 gives the analyses of obsidian and pitchstone.

Table 3 . . . Analyses of Obsidian and Pitchstone[11]

	Obsidian	Pitchstone
SiO_2	73.84	70.19
TiO_2	0.14	0.07
Al_2O_3	13.00	12.37
Fe_2O_3	1.82	1.45
FeO	0.79	0.81
MnO	0.07	0.02
MgO	0.49	0.91
CaO	1.52	1.43
Na_2O	3.82	3.03
K_2O	3.92	3.57
H_2O	0.53	6.48
P_2O_5	0.01	0.03
FeS_2	0.02	
Totals	99.97	100.36

On the basis of the analyses given obsidian contains less than 1 pct, perlite from 3.24 to 4.80 pct, and pitchstone from 4.52 to 10.05 pct water. According to Johannsen the essential difference between obsidian and pitchstone is in the water content, the personal equation entering largely into the naming of the rocks, with obsidians of over 7 pct water and pitchstones of less than 2 pct water having been reported. Johannsen suggests that 3 or 4 pct of water would be a better division point.[12] It would seem, therefore, that texture alone identifies perlite insofar as the rhyolitic glasses are concerned.

The expansion of certain of the rhyolitic glasses when subjected to heat is attributed to the presence of combined water and to some extent seems to be influenced by the release of internal strain. That not all glasses being expanded are perlite is acknowledged, but as a matter of convenience henceforth they shall be referred to as perlite.

Mining

Development of Dant and Russell's Lady Frances mine, in the absence of any data other than that from open pit, surface operations elsewhere, was patterned after standard methods for metal mines. Surface test pits were dug and sampled over a large area. Work was then concentrated on the section giving the best combination of suitable perlite and availability for transportation. The result was the driving of approximately 2000 ft of drifts, crosscuts, and raises whereby approximately 500,000 tons of perlite were blocked out.

During the course of the development work occasional, seemingly unrelated bands of an intrusive resembling rhyolite, locally referred to as "jasperite," were encountered. Dips and strikes along the bands which varied in thickness from a fraction of an inch to several feet were irregular. Many lithophysae were noted on the outer faces of the bands and both the septa thereon and the flow lines within the bands were parallel either with flow lines within the perlite or with cooling fractures within the perlite, and an extremely altered zone of perlite bounded each side of the bands varying from a fraction of an inch to several inches in width. During the initial stages of production it was found that the bands became larger, and more numerous as mining progressed higher into the deposit, and since neither the jasperite nor the adjoining altered perlite would expand, even though the latter ran as high as 8 pct in combined water, it became necessary to sort and discard them insofar as possible. It further was found that the intrusive bands had accompanied post-perlite rhyolite intrusions.

Since fragmentation from blasting usually is very good in perlite the jasperite bands which break coarser are amenable to hand sorting on the basis of size as well as appearance. Currently, sorting of jasperite is accomplished at three places at the mine, all mining now being done by open pit method: (1) fragments too large to be moved by hand are removed from the pit face by bulldozers, (2) fragments too large to pass through the coarse ore bin grizzly are hand sorted by the grizzly man, and (3) fragments too large to pass through the fine grizzly ahead of the crusher partially are picked out by the crusher operator.

That similar rhyolitic bands commonly are associated with perlite deposits elsewhere has been confirmed by J. F. Foran.[a] Mr. Foran commented, however, that not until he had visited the Lady Frances mine did he realize the extent of the bands within the deposit nor the amount of alteration accompanying them.

[a] Formerly Consulting Mining Engineer for Climax Molybdenum Mining Co. Personal communication.

No timbering was found to be necessary during the driving of the 6 by 8 ft development headings, and mining costs were quite low. Drill footage per bit was as high as 162 ft. Five to six 5 ft holes were drilled to pull a minimum 7 ft round. Powder consumption is indicated by the past month's production of 1152 tons of perlite mined with only 250 lb of powder having been used.

During the underground stage of exploration and development, with an eye to the probable necessity at a much later date of mining underground after the surface perlite has been worked out, considerable test work was done in the shrinkage stope and in a later slusher stope to determine the amount of open ground that could be carried without timber. Starting at 8 ft the stopes gradually were widened to 15 ft at which point the back started arching and the sides started sloughing. The maximum permissible width was determined to be 12 ft. Tentative plans for further underground stoping, at such time as that should become necessary, specify sublevel stoping in a series of comparatively small or narrow blocks to facilitate rapid removal of pillars and retreat.

The underground phase at the mine was accompanied by a detailed sampling program. Grab samples were taken of each round blasted and 3 to 4 in. channel samples were cut at 5 ft intervals throughout all the workings, each channel sample giving a minimum of 50 lb of rock to be tested. In addition to the foregoing, special samples were taken of each type of perlite exposed, the types being listed according to textures which varied from true perlitic to splintery to more or less massive, blocky cooling fractures. An interesting observation has been made by Oliver C. Ralston who suggests that the type of cooling fracture may be indicative of the proximity of the center of eruption of the flow.[a]

Table 4 . . . Averages of Combined Water, North Drift

Samples Averaged	Combined Water,* Pct
5—50	3.52
50—145	3.36
145—185	3.23
185—215	3.31

* In running the determination of percentage of combined water the sample is dried at 212°F for 30 min after all apparent moisture has been removed. The sample then is weighed and placed in a furnace at from room temperature to not over 300°F. The temperature gradually is raised to 1200°F at which point it is held for a minimum of 1½ hr. The sample is allowed to cool in a desiccator before final weighing.

Within the groups of samples averaged above, adjacent samples varied as much as 0.26 pct.

[a] Personal communication upon visit to Lady Frances mine, 1946.

Considerable variation was found in the amount of combined water within the deposit. Table 4 lists the averages of water analyses on channel samples cut along the North Drift starting at Station 5 plus 5 ft thence continuing north.

Milling

The milling of perlite is a relatively simple process compared to many of the flowsheets for basic metals or even for other nonmetallics, but, nevertheless, the need for a closely controlled end product, or concentrate, is of extreme importance to the later processing.

As will be explained later the elimination of as many fines as possible from the final expanded product is imperative. Since research has shown that a portion of the fines in the end product are the result of fines (minus 100 mesh material) in the furnace feed or mill concentrate, every effort is made to reduce the fines in the mill. On the other hand, to maintain proper feed control on the furnace as well as to insure the desired screen analysis on the end product, it is just as imperative that there not be an excess of coarse material (plus 10 mesh). Accordingly, from the primary crusher at the mine the perlite goes to storage at the mill thence to a 2 by 6 rod mill for wet grinding at minus 35 pct density. The very friable nature of the perlite due to incipient cooling fractures makes it necessary to remove the perlite as quickly as possible from the grinding circuit once it is down to size to prevent overgrinding.

Samples are taken at half hour intervals throughout the shift and are combined to form composite samples of the day's run. Screen analyses and determinations of percentage of combined water are made on each composite sample. The average of screen and water analyses for the concentrates for the period January 1 to May 15, 1948 is: plus 20, 4.6; plus 60, 73.7; plus 100, 15.4; minus 100, 6.3; percentage of water, 3.16.

During the period of the above average a 20 mesh screen cloth was in use in the mill. At the end of that period a 10 mesh screen cloth was installed which resulted in a marked decrease in the amount of fines as shown in the screen average for May 15 to July 1, 1948: plus 10, 0.3; plus 20, 28.1; plus 60, 55.8; plus 100, 11.1; minus 100, 4.7; percentage of water, 3.26.

A still better average was achieved during August when the minus 100 mesh material dropped to 3.1 pct.

The average life for the medium weight stainless steel screen cloth used is slightly over 3000 tons. The analysis for August 1948, on the screen overflow, was: plus 10, 34.6; plus 20, 53.3; plus 60, 11.5; plus 80, 0.2; plus 100, 0.1; plus 150, 0.1; minus 150, 0.2.

With water being added to the hydroseparator the ratio of water to perlite milled in tons for the entire operation is approximately 15 to 1.

It may be of interest to many operators to note that an average of 55 tons ground every 16 hr is maintained by the 2 by 6 rod mill, normally rated at 1 tph and that rod wear since the beginning of operations has averaged slightly less than 0.5 lb of rod per ton of rock milled. During the first several months of pilot work the wear was considerably higher than the average due to grinding tests at higher densities.

Processing

From the mine at Frieda, Oregon, the crushed perlite is shipped by rail to the processing plants, one at St. Helens, Oregon, another at Grand Rapids, Michigan. There the perlite sand is dried to facilitate handling whereupon it is injected into a horizontal, rotary kiln and subjected to intense heat, approximately equivalent to its melting point. Upon being suddenly brought to this temperature the sand expands to approximately seven times its original volume. From the kiln the popped perlite passes through a series of multicones for cooling and for collecting, dust laden gases passing through a water bath before being expelled and the perlite being drawn into 4 cu ft bags which are sewed ready for shipment or for further processing in the form of ready mixed, fibered, hard wall plaster or ready mixed accoustical plaster. Weight of the multiwall bags of expanded perlite is maintained at 40 to 44 lb per bag, 10 to 11 lb per cu ft.

Heat must be closely controlled during the drying to prevent "burning" of the perlite, removal of an excessive amount of combined water or possible disturbance of the physical character of the rock which renders it useless, resulting in discoloration and excessive weight in the end product. Laboratory tests indicated 600°F as the maximum safe temperature for drying, but actual operation showed that temperature to be too high. Trial and error methods reduced the maximum temperature to 300°F.

A delicately controlled feed is essential. Overloading of the kiln results either in perlite passing through without expanding or, more serious to the operation, in the perlite particles fusing to each other and to the lining of the kiln necessitating costly shutdowns. Underloading, on the other hand, results in loss of valuable production, or, again, in fusion and slagging because of improper dissipation of heat.

Temperature control is of the same importance as the feed control for much the same reasons. Too low a temperature results either in expansion failure or in loss of production. Too high a temperature results in fusion and slagging.

Integrally associated with the feed and temperature are the pressures within the kiln. Rate of flow, the product of the three, is indicated by the fact that within 1 min after the perlite sand has entered the furnace it is in the bag as expanded perlite, having been cooled sufficiently between the actual discharge point on the kiln and the bagger so as not to burn the bags. Perlite currently is processed at the rate of two tons per hour.

Cooling and the all important sizing largely are accomplished by air through a series of multicones between the furnace and the bagger from the first of which is drawn off the standard aggregate and from the second of which is drawn off a product referred to as "cone." Depending on the screen analysis of the standard, a portion of the cone which consists of fines may be added to the standard. The unused portion of cone which amounts to approximately 25 pct of the entire production remains a problem for sales and for research. Except for the small amounts that may be sold for trowel coat in the plastering industry the cone is wasted at present.

Perfect coordination must exist between the fireman and the bagger who, weighing every fifth bag and examining the expanded perlite as it is being bagged, immediately notifies the fireman of changes in weight or indications of burning or fusing in order that proper furnace control can be maintained.

Strength and size specifications for expanded perlite directly reflect the particular market for which it is intended. The plastering field, requiring

the least reprocessing, constitutes the largest market. Consequently in many sections of the United States the perlite aggregate conforms to the ASTM specifications for sizes of sand in plaster aggregate. In the concrete field local building codes dictate the strength requirements, but the demands of the contractors largely determine the particle size. Since the strength and weight are dependent primarily upon the raw material care must be taken in selection of the perlite deposit. It has been noted that perlites having a high water content and fine perlitic texture often expand more than those having a low water content and poor perlitic texture and that the particle strength is inversely proportional to the rate of expansion.

A typical daily average screen analysis of plaster aggregate from the St. Helens plant is shown in Table 5.

Table 5 . . . Screen Analysis of Plaster Aggregate

Cumulative Percent Retained on Screen
August 16, 1948

Mesh	Per Cent	ASTM Specification	
		Maximum	Minimum
+ 4	0.00	0	0
+ 8	0.09	10	0
+ 14	9.32	80	15
+ 28	52.60		
+ 48	79.85	95	70
+100	96.31		95
+200	96.06		
−200	100.00		

Average weight per bag, 41.0 lb.

As improvements were made in the efficiency of the pilot processing plant the amount of salable product became large enough to necessitate a regular sales department which was in operation, therefore, even before the plant had outgrown the pilot stage. Current production of the processing plant already has been stated as being 2 tph. The production figures for the period January 1 to July 1, 1948 are given in Table 6.

The principal market in the northwest, to date, is plaster aggregate. During the first half of 1948 expanded perlite replaced approximately 300 carloads of sand with plaster aggregate and concrete aggregate heading the list of sales. Second on the list was ready mixed accoustical plaster and third, ready mixed hardwall plaster. A noncombustible accoustical tile, still in the pilot plant stage of production, accounted for minor sales.

Outside of the plastering industry, expanded perlite is being used extensively for dry kiln insulation and for roofing insulation slabs. For the dry kilns the slabs are 6 in. thick with a 6 or 7 to 1 mix with portland cement. K factor for the loose aggregate is equivalent to that of ground cork, varying for the slabs according to the amount of binder or cement used. Other applications include soil conditioner, sandtone paint, scouring powder in soap, and filter aides.

Table 6 . . . Production Record for Period January 1 to July 1, 1948

Month	Ore Used (Tons)	Aggregate (Bags)	Cone (Bags)	Total (Bags)	Bags per Ton Raw Perlite
January	349	17,450		17,450	49.9
February	310	14,482	547	15,029	48.4
March	611	24,769	6,472	31,241	51.1
April	414	18,024	3,792	21,816	52.5
May	511	20,997	978	21,975	43.0
June	372	14,192		14,192	38.1
Totals	2,567	109,914	11,789	121,703	
Average					47.1

Summary

With the widespread occurrence of perlite and similar volcanic glass deposits throughout the west and the southwest and with the existence of profitable local markets for the aggregates produced, there is much room for the expansion of processing plants now in operation as well as for the starting of new plants.

New operators can learn much from those already in production, an advantage that most of the present operators did not have when they started. Through underground development and research at the Lady Frances mine in Oregon, it has been found that intrusive, rhyolitic bands, commonly found in perlite deposits, may offer problems in beneficiation or selective mining, perhaps, even appreciably affect the estimated reserves, and that where underground mining is contemplated care must be taken in the choice of mining methods. Specifications for aggregates on the basis of current markets are such that high efficiency must be maintained throughout the milling and processing and care must be shown in the choice of the deposit to be worked.

The research and sales departments must cooperate to create additional markets, whether through new products for existing markets or through new markets for existing products, to reduce the actual loss resulting from unsalable fines.

Acknowledgments

The writer wishes to express his appreciation to Fay W. Libbey, Director of the Oregon State Department of Geology and Mineral Industries for his helpful suggestions as to the phases of Dantore's operation that would be of interest to other operators, to Elko D. Zoradi, Manager of Dantore Products Division for details on the furnace operation and for permission to use data from the Division's semiannual and other production and laboratory reports, and to Peter P. Mackevice and John E. Dwyer, Superintendent and Mill Foreman, respectively, for constructive criticism of the paper and for carrying more than their share of work at the mine during the preparation of the paper.

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10. P. 14 of ref. 7.
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Mechanization at the Bureau of Mines Oil-shale Mine

By E. D. GARDNER* and E. M. SIPPRELLE,† Members AIME

The Synthetic Liquid Fuels Act (58 Stat., 190; 30 U.S.C. Sup., Secs. 321–325) was approved by Congress April 5, 1944; it directed the Bureau of Mines to build demonstration plants to produce synthetic liquid fuels from coal, oil shale, and agricultural products.

The most important oil-shale deposits in the United States are in the Green River formation of Colorado, Utah, and Wyoming. The oil shale of western Colorado generally is more amenable to exploitation, more persistent, and apparently richer than elsewhere in the Rocky Mountain Region. It occurs at the top of a high plateau surrounded by bold, nearly vertical escarpments 500 to 600 ft in height. The top 400 to 500 ft of these escarpments comprises an oil-shale measure that averages 15 gal of shale oil per ton; the bottom 70 to 100 ft of the measure, called the Mahogany Ledge, averages over 30 gal per ton. The full measure in a 1000-square-mile area is estimated to contain 300 billion barrels of shale oil;[1] the Mahogany Ledge is estimated to contain 100 billion barrels of oil. These estimates are based on numerous surface samples and on core-drill holes drilled by the Government and by private enterprise.

The oil-shale beds are undisturbed and lie nearly horizontal. The name "oil shale" is unfortunate, as the rock is a tough, strong marlstone. Organic matter named "kerogen" is a constituent of the rock.

The oil-shale demonstration mine is on Naval Oil-Shale Reserve No. 1; it is about 5½ miles by mountain road from the plant site, which, in turn, is 2 miles from U.S. Highway 6 and 10 miles west from Rifle, Colo. The highway parallels the Colorado River, the Denver and Rio Grande Western Railroad, and a 66,000-volt public-utility power line.

Early in the program it was realized that an oil-shale enterprise would have to be on a large scale to be commercial, and that unusually low mining costs would be necessary. The physical characteristics of the Mahogany Ledge and an overlying roof stone are unusually favorable both for large-scale operation and low mining costs. Quarry practices largely will be followed underground to mine the Mahogany Ledge. The aim has been toward complete mechanization of all mining operations.

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A mining unit would comprise a square mile containing 100,000,000 tons of oil shale with allowance for mining losses. An investment of $1,000,000 would be $0.01 a ton; such an expenditure, therefore, could be made to save, say, $0.015 per ton daily operating cost.

A goal of a mining cost of $0.50 per ton was set up in 1945. Open-cut mining costs then were about $0.25 per ton of material handled; it was hoped that a cost double this amount could be obtained in mining the Mahogany Ledge. Costs of labor and supplies have increased since 1945 and so has the selling price of petroleum products.

To those unacquainted with the character of the Green River deposits, the name "shale" suggested high mining costs. Others assumed that coal-mining costs of $2 to $5 per ton would apply. To many, the expected mining costs appeared the principal handicap to the establishment of an oil-shale industry. The purpose of the Bureau of Mines is to demonstrate methods and to establish costs for mining the oil shale on a large scale. It is hoped this work will lead to the establishment of a large-scale, commercial oil-shale enterprise.

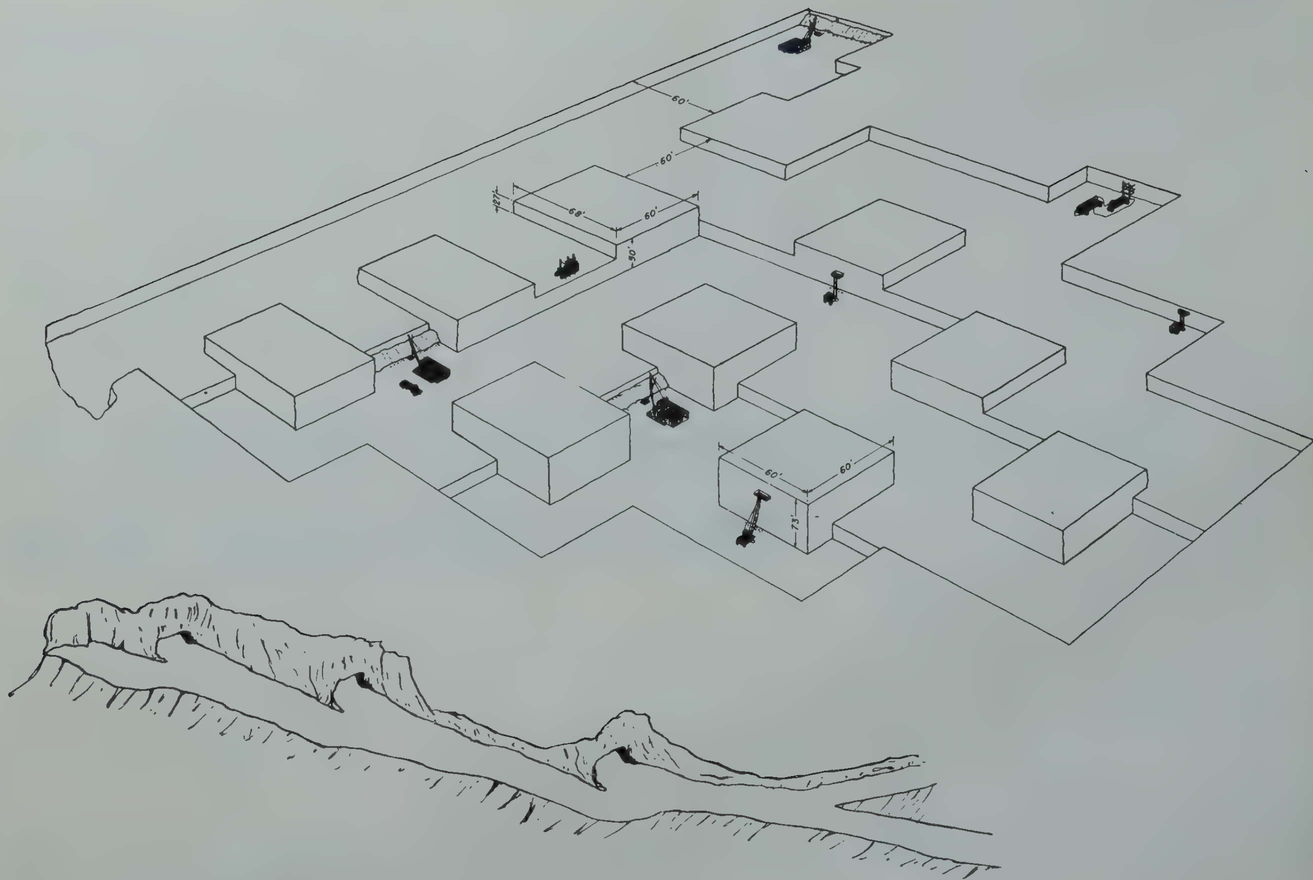

FIG 1—Underground quarry mine layout, oil-shale demonstration plant, Rifle, Colo.

Mining Methods

The first problem to be solved, obviously, was to determine how large openings could be maintained safely underground without artificial support. To this end, representative samples of the Mahogany Ledge and of the roof stone were tested at Columbia University and at the Bureau of Mines stations at College Park, Md., and at Pittsburgh, Pa.[2] The theoretical results indicated that the overlying formation could be supported by 60 ft square pillars spaced 60 ft apart, and that the roof stone would safely stand in the openings. A test room 50 by 100 ft was excavated under the roof stone in 1946 to obtain a practical check on the theoretical results. It was widened to 60 ft in May 1947, to 70 ft in August 1947, and to 80 ft in November 1948. During this period, daily sag measurements were taken; rock noises also were recorded through a system of geophones connected to a microseismic recorder. To date there has been no evidence that undue stresses have developed in the roof stone either in the test room or the mine workings. The danger of projecting the results beyond the workings is recognized. The quality of the roof stone, however, is so remarkably consistent that it is felt the data are reliable.

It was decided to mine the Mahogany Ledge by advancing three levels horizontally into the Mahogany Ledge. The advance heading is 27 ft, and each of the other two levels is 21½ ft high; the lower levels will follow the advance of the top level similar to benches in a quarry. Fig 1 illustrates the proposed mine layout. By having the advance level at the top, work on all three levels is under the good roof stone. Moreover, the roof and upper parts of the pillars are more accessible for initial scaling and inspection from the top level than from a lower one.

Breaking

The only major operation in the underground mine essentially different from quarry practice is drilling and blasting in the advance heading. Faces on this level have only one free face to which to break. Drill holes must be drilled more accurately to pattern, more holes are required, and more explosive is needed per ton broken than will be required on the lower benches, which will have two free faces. Moreover, drilling horizontal holes in a 27 by 60 ft face is much more difficult than drilling vertical down holes on a bench.

To date most of the effort at the oil-shale demonstration mine has been the development of equipment and procedures for drilling and blasting on the top level.

DRILLING

The initial drilling investigations were time studies of conventional drilling practices using standard 3 in. rock drills mounted on columns. The results obtained from these studies were startling; a miner and chuck tender were required for each machine, and the drill was actually drilling less than one-third of the work shift. The remainder of the time was spent in setting up, tearing down, changing drill steel, changing position of the drill on the arm and column, and other tasks.

Analysis of these data indicated that the drilling phase was a logical one with which to start mechanization. It was reasoned that if changing drill steel could be eliminated by the use of 10

to 15 ft drill rods and of 10 to 15 ft drill feed carriages, and if the setting up, dismantling, and changing of the drill position could be mechanized adequately, at least one man of the drilling crew per machine would not be needed. Drilling with long rods in a mechanized operation also would allow a greater actual drilling time per machine.

The first obstacle in the path of the program for the mechanization of drilling was that conventional detachable rock bits could drill only from 3 to 5 ft in oil shale without becoming dull. This, of course, required changing of bits or rods several times for each hole. To overcome this handicap, a research program was instigated to develop a hard-surfaced rock-drill bit that would drill at least 10 ft without becoming dull. As a result, a bit hard-surfaced with tungsten-carbide was developed, which consistently drills over 70 ft before it requires resharpening.[3]

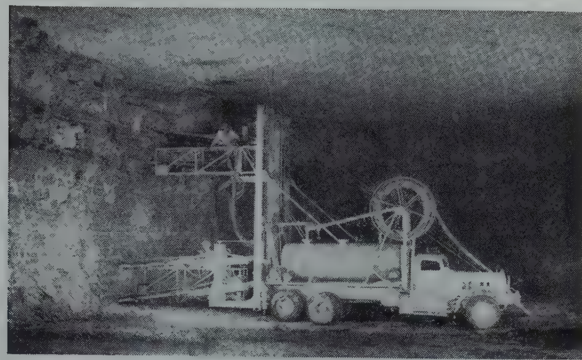

FIG 2—Multiple drill carriage or jumbo for drilling advance heading of underground quarry.

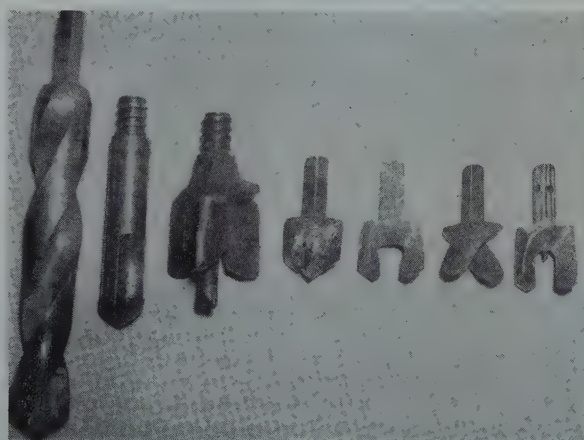

FIG 3—Experimental auger-type bits for drilling oil shale. Left to right: 1⅞ in. carbon steel twist drill with hard-surfaced cutting edges; 1½ in. insert-type Cyclone masonry bit; 3½ in., 3-prong Kennametal insert bit with shop-made pilot; 2½ in., 2-prong Sulmet insert bit with shop-made pilot; 2¼ in., 2-prong standard Kennametal insert bit without pilot or core breaker; 2¼ in., 2-prong field insert bit; 1⅞ in., 2-prong Kennametal insert bit with core breaker.

Mechanizing the various drilling operations for drilling flat holes of a round in a 27 ft high by 60 ft wide face presented another problem. A solution for this was found in a new and efficient type of multiple-drill carriage (or jumbo) designed and built on contract (Fig 2). This unit has two horizontal platforms mounted on a framework at the rear of a diesel truck. Each platform contains two 4 in. percussion drills mounted on 11½ ft feed slides. The vertical inclinations of the feed slides are controlled by screws and ratchets. Air hoists are used for raising and lowering the platform; spur gear devices permit swinging the platforms to various horizontal angles. A water tank and air hose reel also are mounted on the truck. Only the 3-in. air hose has to be connected to the air supply preparatory to drilling. The four drills mounted on the carriage are operated by two men. By use of the multiple-drill carriage, two miners can drill a round with 10 ft steel in about 5 hr. Prior to the time the Bureau of Mines multiple-drill carriage was placed in operation, drilling on the top level was done with conventional wagon drills which required two men to a machine. Over 10 man-shifts were required to drill a 10 ft round with this equipment. Longer drill feed slides have been designed and are being built for the multiple-drill carriage. These slides will enable 15 ft holes to be drilled. It is expected that these longer rounds can still be drilled by two men in an 8 hr shift and will break 1500 tons.

Although breakage of the long drill rods because of fatigue appears about normal, the cost of steel per ton of oil shale mined is excessive. A research problem was set up to investigate the use of alloy drill steels and to find some method of treating used drill steel to relieve fatigue strains. Experiments using alloy drill steel have been discouraging, and no furnace long enough has been found in which to treat the drill steel. An encouraging result was noted while testing an Ingersoll-Rand type 2 insert bit. The life of the drill steel during this experiment was almost double that obtained with another hard-surfaced bit. The increased life is believed to be due to the fact that the insert bit maintained a sharper cutting edge throughout the life of the rod, thereby reducing the fatigue shock. At the time of writing, the bit had drilled 1680 ft in oil shale with no apparent damage or dulling.

Although the mechanized jumbo using percussion drills powered with compressed air may be considered adequate for the drilling phase of the top heading, it is not necessarily the best solution. A research program currently is being conducted at Rifle to develop a rotary drill and auger-type bits for drilling oil shale. The rotary drill has a number of advantages over the percussion drill:

1. The drilling rate is higher.
2. The breakage of drill rods would be considerably less.
3. Electric power could be substituted for more costly compressed air.
4. Deeper rounds could be drilled without loss of drilling speed.
5. The noise of percussion drills would be eliminated in the mine workings.

As in the case of percussion drilling, the selection and development of a bit is the primary problem. Numerous types of rotary drill bits with tungsten-carbide inserts are being tested. Some of these bits are shown in Fig 3. Nearly all of the bits pictured have satisfactorily drilled the higher grade oil-shale

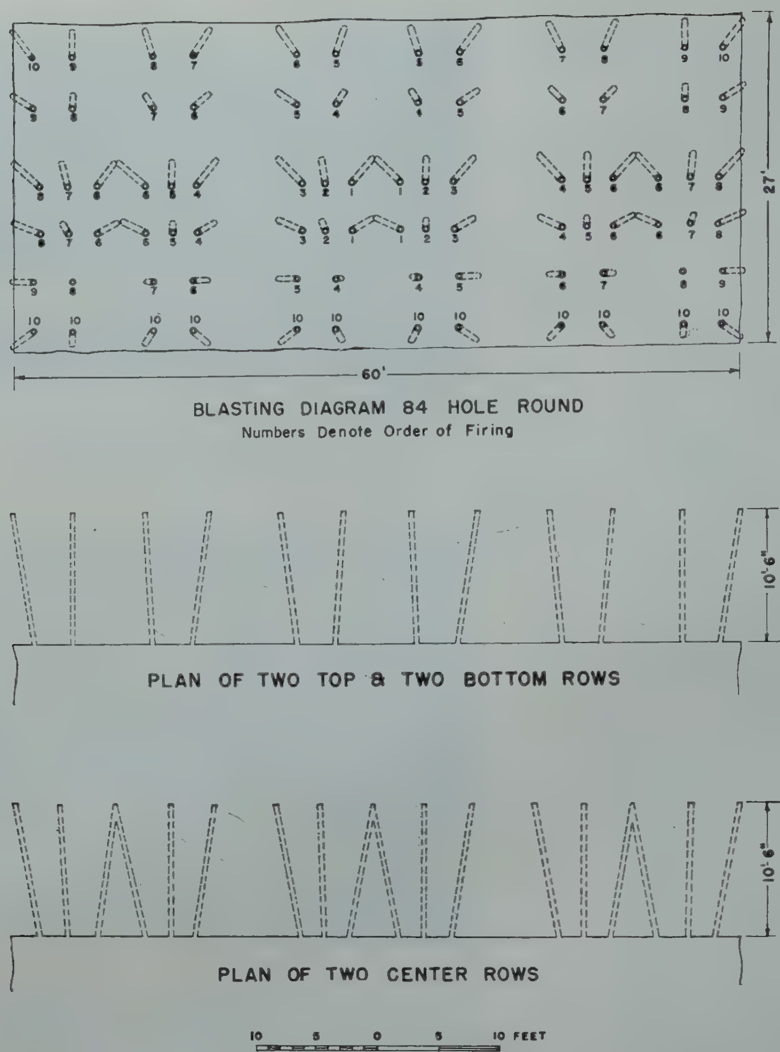

FIG 4—84 hole round using 3 V-cuts.

beds of the middle third of the Mahogany Ledge at a rate of 36 to 42 in. per minute actual drilling time. As the inserts either become dull or broken with 3 to 5 ft of drilling in the lower-grade oil shales of two-thirds of the Ledge, the bits cannot be considered adequate. One possible exception is the Cyclone masonry bit manufactured by the New England Carbide Tool Co. This bit, in the one test made up to the time of writing, drilled 70 ft into the lower-grade oil shale at a rate of 33.5 in. per minute before resharpening was necessary. A bit is desired that will have a life long enough to compensate for the higher cost of this type of bit; it also should have a drilling speed, with a rotary drill, at least double that ordinarily obtained with the percussion drills.

Many standard rounds for large headings, with variations, have been tried. The pattern found to be most successful to date in a 60 ft wide by 27 ft high heading is shown in Fig 4. The top holes bottom about 1 ft from the roof stone and break to an uncemented bedding plane at the roof stone. This round has a total of 84 holes averaging 10 ft in depth; it will break about 1000 tons of rock with a powder consumption of 0.56 lb per ton. It is drilled in three settings of the jumbo and is blasted in three series in a parallel. This round will be used as the basis for developing the best pattern for a 15 ft deep round that will be drilled when the new slides are installed on the multiple-drill carriage.

BLASTING

The cost of explosives probably will be the largest single item of expense in mining oil shale on a large scale. Considerable research, therefore, is justified to ascertain the grade and type of explosive that will give optimum results and to determine the best procedures to follow for a minimum overall breaking cost. The goal set up at the beginning of the project was not to exceed half a pound of explosive per ton of oil shale broken.

Early blasting experiments were made with du Pont Gelex No. 2, 45 pct strength; Gelex No. 3, 40 pct strength; and Extra C and Extra C-1 explosives. These tests indicated that good fragmentation is obtained with the Gelex No. 2, 45 pct strength, with the smallest consumption of powder per ton of rock broken. Further tests are to be made, but this explosive has been adopted as a constant in current experimental work with heading rounds.

Charging a round in a 27 by 60 ft heading such as described above is time-consuming. Stagings were required, and handling the explosives to the stagings was difficult and hazardous. A special piece of equipment was designed and built from which to charge the blastholes. This unit con-

sists of a wooden, 5 by 10 ft platform mounted on a Wagnermobile fork-lift truck (Fig 5). The truck is powered by a diesel engine and has a power hoist for raising and lowering the platform from which the miners charge the explosive into the drill holes. In addition, there is a ladderway up the side of the hoist frame and a remote-control cable to permit lowering of the platform by the miners working on it. The man-hours required for blasting have been reduced by about one-third through the use of the unit. At present, two miners can charge and blast a round in 5 hr. The rounds are loaded with three settings of the rig. The platform is of adequate size to allow carrying enough explosive to charge all the holes for each setting, thereby simplifying the explosives-handling problem.

No adequate or safe mechanized process could be found for tamping explosive into the blastholes. Experiments in which only the last cartridge in each hole was tamped indicated no apparent loss in blasting efficiency. At present, 1¾ by 8 in. cartridges are used in 1⅞ or 2 in. constant-diameter blastholes, and only the last cartridge in each hole is tamped.

Considerable experimental work remains to be done to perfect the drill pattern, to ascertain the optimum quantity and the best grade and type of explosive, to determine the best sequence of firing the blastholes, and to find the optimum ratio of diameter of drill hole to length of column of explosive in the hole.

FIG 5—Mobile blaster's platform from which to charge blastholes in advance heading of the underground quarry.

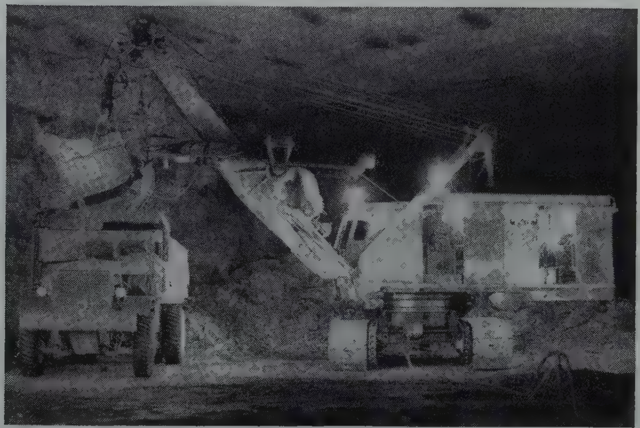

FIG 6—A 3 cu yd electric shovel loading a 15-ton Euclid dump truck on advance heading of underground quarry.

Loading and Transportation

Preliminary studies indicated that a continuous-type loader discharging directly onto a conveyor belt, which, in turn, would transport the broken shale to the crushing plant, would be the most efficient ultimate practice of handling the broken oil shale. This procedure was not adopted, because no standard continuous-type loading equipment capable of loading the large fragments of broken oil shale (up to 5 ft in diameter) is made, nor are conveyor belts made to handle such material. It was then necessary to accept standard loading and transportation equipment such as is used in quarries.

A standard 2½ yard, Bucyrus-Erie electric shovel with a special short boom to permit working under a 25 ft ceiling was chosen for loading the broken shale. As the shale is relatively lightweight (16 cu ft to the ton in place), the 2½ yard dipper was replaced with one of 3 cubic yards capacity. Two 15-ton Euclid diesel end-dump trucks were obtained from the Navy, and later a third truck of the same size was purchased for transporting the shale. Fig 6 shows the electric shovel loading a dump truck. The electric shovel appears to be the largest unit that can work efficiently in the 27 ft high by 60 ft wide openings; a larger dump truck, however, might be more efficient for transporting the oil shale.

The question always is raised as to the advisability of using diesel equipment underground. It follows that a part of the program at Rifle is to demonstrate under what conditions diesel equipment may be used in large underground workings.

The objectionable constituents of exhaust fumes from a diesel engine are carbon monoxide, oxides of nitrogen, and aldehydes. Carbon monoxide and the oxides of nitrogen are toxic gases. The formation of toxic gases may be kept to a minimum in diesel-engine exhausts by proper adjustments to allow a maximum amount of intake air per pound of fuel used. It has been found that the exhaust gases are diffused in the atmosphere of the large mine openings and that harmful concentration of the toxic gases does not occur.

The aldehydes, although not particularly harmful, have a characteristic sharp, suffocating odor and, even in small quantities, cause irritation to the eyes and respiratory passages. During

FIG 7—Portable scaling rig from which to scale loose rock from roof and pillar walls on advance heading of underground quarry.

the period before mechanical ventilation was installed at the oil-shale mine and when there was no natural movement of air (the outside and inside temperatures being the same), concentration of aldehydes from the diesel trucks in a working place became intolerable. These conditions exist only during short periods in the spring and the fall; during the remainder of the year natural ventilation is adequate.

Ample and adequate ventilation is requisite for the use of diesel equipment underground. It has been recommended that the minimum volume of ventilating air required for operating diesels underground is about 75 cfm per brake power.[4]

The elimination of the aldehydes from the exhausts of diesel engines to be used underground would be desirable even where ventilation is good. Currently, two methods to remove aldehydes from diesel exhausts are being investigated at the Rifle mine. One of the methods is to scrub the exhaust gas with a solution of sodium sulphite in water, using hydroquinone as an inhibitor.[5] The other method is to pass the exhaust fumes through a catalyst to remove the aldehydes. This latter system was developed by the Catalyst Research Corp. of Baltimore, Md.

Scaling

In any underground mine it is necessary to inspect the roof stone and walls and pry off any loose rock that might fall and cause injury to personnel or damage to equipment. At the oil-shale mine, where the pillars will eventually extend 70 ft above the floor, it is especially essential. Scaling is an important item of cost, and special equipment is required, so that the work may be done efficiently. A scaling rig (Fig 7) was acquired for use on the top level; it comprises a traveling crane equipped with a self-leveling platform, according to a design made at the oil-shale mine. Two men working on the platform remove loose rock from the roof and the upper parts of pillars by means of aluminum rods with steel tips. A small percussion drill can be mounted on the platform for drilling loose slabs of rock not easily pried off. Similar equipment capable of reaching a greater height is to be obtained as the lower levels are advanced

Ventilation

Forced ventilation is supplied by a 60,000 cfm, low-pressure fan. Doorways have been built in each of the two entrance adits to the mine. The fan is installed in a 20 sq ft opening in a door; it can be moved from level to level, depending upon which level is being worked.

Facilities

POWER

Electric power is supplied in the mine at 2300, 440, and 110 volts, the 2300-volt circuit being used for the shovel. Several floodlights are installed on each piece of mobile equipment to ensure adequate lighting of the working places. In addition, permanent lights are suspended from the ceiling along the main haulage opening and at all plug-in stations.

The power lines are buried in trenches from the main substation into the working area. Throughout the working area, they are carried along the roof stone with drops to plug-in stations at convenient locations on each pillar.

WATER AND COMPRESSED AIR

Calculations made during initial operations showed that the cost of burying and moving air and water lines on the advancing levels of a commercial oil-shale mine would be an important item of expense. Moreover, loss of air pressure because of long air lines would reduce drilling speed with pneumatic drills, and the air and water lines would interfere with other work. A movable compressor assembly has been designed and is under construction. This unit will have two 750 cu ft electric air compressors, an air receiver, and a water tank. Air and water lines will not be required with this equipment, with a consequent saving in operating costs. The compressor motors will take the same voltage as the shovel, and the same outlets may be used interchangeably between compressor and shovel. A compressor and a shovel would not be working in the same heading at the same time.

Water for the experimental mine is piped in from a reservoir on the mesa. Water in a commercial mine would be trucked to where needed for drilling and wetting down broken piles of rock.

Test Run No. 1

The essential equipment to conduct a unit mining operation had been assembled by August 1948. A 20-day test run was made in August and September; each phase of the operation was completed in a single shift each day. Two 60 ft and two 50 ft headings were available for the test. The test was conducted to demonstrate the unit production of each phase of the operation, to obtain operating and cost data, and to assemble information that might be helpful in improving mining technique.

A total of 10 miners was assigned to work underground. Two miners on the jumbo drilled a 10 ft round each shift. Two miners in another heading loaded and blasted the round previously drilled. The broken shale in a third heading was loaded by one shovel operator and transported one-third of a mile to a stockpile by two truck drivers; a bulldozer operator cleaned up fly rock. The face of a fourth heading was scaled preparatory to drilling by a 2-man scaling crew. Two engineers and a statistical clerk were assigned full time to the test as bosses and to take and record pertinent data.

During the test run an average of 840 tons was mined each shift for a total production of 16,800 tons. Production per man-shift for underground labor was 81 tons, or 56 tons per man-shift total labor. The operating cost was $8,352, or $0.497 per ton.

The tons that could be loaded in a

Table 1 . . . Summary of Costs—First Test Run in Underground Quarry

	Labor[a]	Power	Fuel	Explosives	Other Supplies	Maintenance	Totals
Direct operating costs:							
Drilling	$0.028	$0.020			$0.054	$0.016	$0.118
Blasting	0.037			$0.106		0.002	0.145
Loading	0.032	0.005	$0.006			0.015	0.058
Transportation	0.029		0.007		0.008	0.005	0.049
Scaling	0.025		0.001		0.001	0.003	0.030
Direct supervision	0.042						0.042
Total	0.193	0.025	0.014	0.106	0.063	0.041	0.442
Indirect operating costs:[b]							
Engineering	0.007						0.007
Miscellaneous	0.005		0.001		0.002		0.008
Leave and vacation pay; 17.1 pct payroll	0.040						0.040
Total	0.052		0.001		0.002		0.055
Totals	0.245	0.025	0.015	0.106	0.065	0.041	0.497

[a] Some labor included in "Maintenance."
[b] Does not include general supervision, office expense, depreciation, or interest on investment.

shift by the shovel has been the unit to which other phases of the mining were to conform. The test indicated that this capacity would be 1400 to 1500 tons. It was demonstrated that the drilling and blasting phases could be completed in 5 hr. It would appear that by the addition of the longer slide on the jumbo, a round to break this tonnage could be drilled in a shift; the longer round also could be charged with explosive in a shift by two men. The overall tons per man-shift underground thereby would be increased nearly 50 pct.

The summary of costs is shown in Table 1, and a summary of engineering data is shown in Table 2.

Table 2 . . . Test Run No. 1—Engineering Data

Labor Item	Tons per 8-hr Man-shift: Underground	Total
Drilling	425	300
Blasting	354	337
Loading	382	306
Transportation	416	352
Scaling	458	440
Direct supervision	420	420
Engineering	2,440	2,440
Total force	81	56

Power and supplies		
Power kw-hr per ton:		
Drilling	0.85	
Loading	0.27	
Utilities	0.13	
Total		1.25
Explosives pounds per ton broken		0.56
Tons broken per foot hole		0.84
Feet holes drilled per drill bit		78.30
Tons per pound drill steel broken		6.50
Tons per load hauled		16.25
Tons hauled per gallon fuel		40.20

The highest single item of expense was $0.054 for drilling supplies; of this amount, nearly $0.04 cents was for broken drill rods. It is hoped that investigations now under way will bring this cost of drill steel down to not over $0.02 per ton of oil shale broken.

The labor cost of drilling ($0.028) and of charging the drill rounds ($0.037) will be reduced by drilling longer rounds. The cost of explosives was $0.085 per ton. Recent improvements in the distribution of explosive in the drill holes indicate that a round can be broken with less explosive; the saving in cost may be $0.01 per ton.

The labor cost of loading was $0.032. The shovel has a greater capacity than was utilized in the test run. When larger tonnages are handled, the labor cost will be reduced. The labor cost for transportation, however, probably will remain the same ($0.044).

During the test run, the full time of two scalers was required to prepare the drilling faces. There was not enough time to inspect adequately the roof and pillars in the rest of the mine. Although longer rounds will reduce the amount of face scaling per ton mined, the present cost of 2.6 cents a ton probably is low.

The results of the test run were encouraging; they show that the original goal is possible.

Summary

Among more important accomplishments to date are:

1. Determination of room-and-pillar dimensions for mining the Mahogany Ledge by an underground method.
2. Selection and development of a large-scale mining method for demonstration of low-cost mechanized mining.
3. Development of a hard-surfaced rock bit for drilling over 70 ft in oil shale before resharpening is necessary.
4. Design and development of a four-machine, multiple-drill carriage with which two miners can drill an 84-hole round 10 to 15 ft deep in one shift.
5. Design and procurement of a fork-lift truck from which to charge blastholes on the top level, and development of a charging technique for reducing charging time.
6. Selection of an optimum size electric shovel for loading broken oil shale and equipment for transporting it to the stockpile.
7. Design and procurement of a self-leveling platform on a tractor-mounted crane from which to scale loose rock from roof and pillar walls.
8. Demonstration, by means of an actual production test run, of equipment performance, mining costs, and mining technique.

The usual number of failures have occurred. About a year and a half was spent in the design and fabrication of a first jumbo; it proved a failure in one shift. The same general design, however, was used for the second jumbo, which was used in the test run. Moreover, the second jumbo had to be modified for the use of 15 ft slides. The effect upon costs of breakage of the 10 and 15 ft drill rods was overlooked until after drilling of long rounds was commenced.

Much work remains to be done in perfecting procedures for drilling and blasting the top heading.

Efforts to date have been confined largely to the top level. Drilling and blasting procedures for breaking the lower benches are yet to be developed. An important point now under discussion is to determine whether the part of the Mahogany Ledge which is below the top level is to be mined in one or two benches.

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Enhancement and Hazard Factors as Related to Mine Valuation

By J. MURRAY RIDDELL,* Member AIME

Introduction

The method of treating hazards wherein value is decreased, is cited by R. D. Parks.† Quite properly, the theory of probabilities is made use of when multiple hazards are under consideration. E. F. Fitzhugh, Jr.‡ deals with the enhancement phase graphically and uses percentage of expectancy as derived by a study of conditions and the selection of a chance factor. It is believed that both of the authors referred to fully recognize the part which human judgment plays, and as coupled with experience, training, and the application of the theory of probabilities; in that respect the composer of this paper is in accord.

Within the past year, the task of arriving at a mathematical method for handling the enhancement phase, and, additionally, coupling the negative and positive phases, was explored. In citing the hazard, or negative phase, reference is made to the text by Parks.

Types of Items for Consideration

On the hazard, or negative, side some of the common items to be considered are:

1. Subsidence and resulting damage.
2. Flowage of water above that anticipated.
3. Geological.
   - (a) Decline in quality of probable ore, and/or
   - (b) Decline in quantity of probable ore.
4. Depressed economic conditions.
5. Litigation.

Similarly, on the enhancement, or positive, side common items to be considered are:

1. Flowage of water below that estimated.
2. Geological.
   - (a) Increase in quality of probable ore, and/or
   - (b) Increase in quantity of probable ore.
3. Elevated economic conditions.
4. Litigation.
5. Technological advance in the art.

In all of the above items no consideration is given to placing a value on, or giving consideration to, possibilities.

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† R. D. Parks: Examination and Valuation of Mineral Property. Cambridge, Mass. Addison-Wesley Press, Inc.

‡ Edward F. Fitzhugh, Jr.: The Appraisal of Ore Expectancies. TP 2090, *Min. Tech.* (Jan. 1947); *Trans.* AIME (1948) 178, 143.

Hazard Phase

For example, referring to Table 1, the estimated normal present value (Vp) of a property, exclusive of risk consideration, is taken as \$100,000. If subsidence affects the usefulness of the shaft, the present value of the property will decrease to \$60,000, and it is judged that there is a 1 in 4 chance that the shaft will be impaired; consequently, if subsidence took place, the present value would be reduced (\$100,000–\$60,000) \$40,000, but since the chance is 1 in 4, the allowable reduction would be only one-fourth of \$40,000, or \$10,000, or a discount of 10 pct (hazard factor) of the original value; the related safety factor is taken as unity minus 0.10, or 0.90. Thus, the original Vp, \$100,000, multiplied by the safety factor (\$100,000 $\times$ 0.90) gives \$90,000, which is the adjusted present value of the property in this single instance.

Additionally, there is a water hazard, which, if encountered, will reduce the present value to \$85,000. The chance is 1 in 3. Following through with the same reasoning as shown above, the derived safety factor will be 0.95.

To arrive at a single multiplying factor which will express the combined risks, the two safety factors are multiplied together, $0.90 \times 0.95 = 0.855$. That factor when multiplied by the original Vp will give the adjusted, and

Table 1 . . . Hazard Phase

Present value of property, exclusive of risk consideration, estimated, Vp = $100,000

Nature of Risk	Chance	Potential Value	Potential Reduction A	Allowable Reduction B	Hazard Factor C	Safety Factor D
Subsidence	1 in 4	$60,000	$40,000	$10,000	0.10	0.90
Water	1 in 3	85,000	15,000	5,000	0.05	0.95

Combined safety factors = 0.90 × 0.95 = 0.855

Adjusted present value = $100,000 × 0.855 = $85,500

Note: A = Vp − potential value

B = A × chance ratio

C = B ÷ Vp

D = unity − C

Both may happen,	0.10 × 0.05 = 0.005
Subsidence may happen, water may not happen,	0.10 × 0.95 = 0.095
Water may happen, subsidence may not happen,	0.05 × 0.90 = 0.045
Both may not happen,	0.90 × 0.95 = 0.855
	1.000

Table 2 . . . Enhancement Phase

Present value of property, exclusive of enhancement consideration, estimated, Vp = $100,000

Nature of Risk	Chance	Potential Value	Potential Increase	Allowable Increase	Security Factor	Improbability Factor
Geological	1 in 6	$175,000	$75,000	$12,500	0.125	0.875
Technological Advance	1 in 2	140,000	40,000	20,000	0.200	0.800

Combined improbability factors = 0.875 × 0.800 = 0.700

Multiplying factor = (1 − 0.700) + 1.000 = 1.300

Adjusted present value = $100,000 × 1.300 = $130,000

Both may happen,	0.125 × 0.200 = 0.025
Geological may happen, technological may not happen,	0.125 × 0.800 = 0.100
Technological may happen, geological may not happen,	0.200 × 0.875 = 0.175
Both may not happen,	0.875 × 0.800 = 0.700
	1.000

weighted, value of the property, or $85,500. The procedure is the application of the theory of probabilities to the problem.

Enhancement Phase

Referring to Table 2, the original Vp remains the same as previously quoted, $100,000. The involved enhancing phases under consideration are two, namely: (1) probable ore extension, geological; and (2) the probability of technological development which would elevate value. In the first case the chance is considered as 1 in 6, and in the latter 1 in 2; correspondingly, the respective enhanced present values are taken as $175,000 and $140,000.

Enhancement and improbability factors are derived in a manner like the hazard and safety factors of Table 1.

It will be noted that the multiplying factor is expressed as (1 − 0.70) + 1 = 1.30; that raises the question, why? Again it is a play on the theory of probabilities.

Since the improbable portion of the last entry is 0.700, then the probable portion is 1.000 minus 0.700, or 0.300; further, it is then necessary to add unity, 1.000, in order to care for the base figure.

Combination of Hazard and Enhancement Phases

Should a condition exist where both phases, hazard and enhancement, surround a property, and wherein a final adjusted present worth figure is desired, such can be obtained by combining, by multiplication, the original present worth and the respective multiplying factors, thus:

Final adjusted present value = $100,000 × 0.855 × 1.300 = $111,150.

Conclusions

It is believed that the foregoing method of caring for situations of the kind described is basically sound. However, it is recognized that the selection of the chance factor is a matter of personal judgment based on experience, observation, and deduction; it could well be the average of the judgment of two or more appraisers.

The foregoing example has made no mention of formula to be used in arriving at the respective present values under given conditions, nor rates of interest to be considered, because they are outside the premise of the subject matter.

The Economics of Geophysics in Mining Exploration

By J. J. JAKOSKY,* Member AIME

The strategic importance of the metallic minerals in our industrial economy, and the declining rates of discovery have focused attention on means of exploration for new mineral deposits. A consideration of exploration techniques leads to an attempt to evaluate the effectiveness of geophysics as a tool for mining exploration. One means of evaluation will be to compare the use of geophysics in petroleum exploration with its possible uses in mining.

Geophysics in Petroleum Exploration

During the past twenty years, the petroleum industry has established an effective exploration technique. Geology is the basic tool in those areas where reliable predictions can be made from surface conditions. In the other areas, where the geological studies are inadequate, they must be supplemented with geophysics or direct exploration, such as drilling. Since geophysical work is far less costly than any form of direct exploration, it is used almost exclusively as the means for obtaining the desired subsurface information.

The geological and geophysical techniques are credited with locating sites which yielded one producer for three dry holes during the year 1947.[1] For the wells located without geology or geophysics, there is a ratio of one producing well to sixteen dry holes. From these figures it is seen that the geological and geophysical techniques are more than four times as successful as the nontechnical exploration.

It will be in order to consider further this ratio of one producer to three dry holes. The geophysical methods in use today† do not pretend to locate oil directly; their function is to locate anomalies or structures favorable for the accumulation of oil. For a producing oil well we must have not only structure, but also other favorable conditions, namely: petroliferous source beds, the required porosity and permeability in the reservoir rocks, an impermeable cap rock over the reservoir, and the necessary water or gravity drive to force the oil into the bore hole. Since the geophysical work can only detect and sometimes delimit the anomaly, these other necessary conditions can be determined only by direct exploration. The success of modern geophysical work in locating and limiting the structure itself is remarkably high, probably over 95 pct accurate.

Geophysical work renders an important service in the delineation of the structurally unfavorable areas and their elimination from the drilling phase of the exploration program. When attempting to locate any material of scarce occurrence in the earth, it is of paramount importance to know where *not* to dig. This elimination of barren areas is the least appreciated but is of great economic importance. For illustration, the difference between one producer to three dry holes, and one producer to sixteen dry holes, represents a saving of thirteen drilled wells, which would require an expenditure many times the cost of the geophysical work that might have revealed the nonfavorable structural conditions.

The percentage of geophysically-located wells necessarily will continue to increase in the statistics of future years, as the search for oil is extended

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[1] References are at the end of the paper.

† Excluding the geochemical method, which is still undergoing development.

into the areas where more geophysics must be employed. In the United States, the most important undeveloped petroliferous areas remaining today are the covered or masked areas, complex fault accumulations, and the stratigraphic traps created by unconformities, pinch-outs, overlaps, changes in porosity or sedimentation, and lensing. These areas generally are not amenable to geological studies, and must be attacked by geophysical work or direct drilling.[2]

The high trend in the use of geophysics for petroleum exploration is increasing in spite of the increased costs occasioned by more intensive and detailed work. During 1947, over $90,000,000 was expended for seismic work. During this same period, the expenditures for gravity work totaled about $7,500,000. Probably an additional $7,500,000 was spent on the air-borne magnetic, electrical, geochemical, and other methods. The total expenditure for petroleum geophysics last year was therefore in the neighborhood of $105,000,000.[3] During this same period, the value of the crude oil, natural gas, and natural gasoline produced was in excess of 2.5 billion dollars. This means that today the petroleum industry is spending about 4.2 pct of the gross value of the oil produced on direct geophysical work, and perhaps another 5 to 10 times that amount on geological and other exploration necessary to find new oil. Although this large expenditure is made for technical guidance, the history of the industry clearly shows that the greater success factor of the technically-located wells more than pays for this cost by the savings effected in not drilling the unfavorable areas.

Now let us examine the use of geophysics in mining. The application of geophysics to mining exploration is so spasmodic and spotted that we have no reliable scout reports showing the number of crews operating. The best information that I have been able to obtain comes from the state mining bureaus, consulting organizations, and private mining companies. This information indicates that about $450,000 per year will cover the entire expenditures for actual field work and applied research. During that period, the annual value of the metals mined was about $1,800,000,000. This gives an expenditure for mining geophysics of about 1/40 of 1 pct. The petroleum industry is therefore spending a percentage that is more than 165 times as large as that being spent by the metal mining industry.

Exploration Procedures in Petroleum and Mining Exploration

There is a fundamental reason why geophysical work has proved to be such an important tool in petroleum exploration, and why as yet it is so relatively unimportant in mining. This reason is one of procedure, based more on mining precedent than on technology or economics. Over a period of years, the petroleum explorer has learned that the best way to discover a new oil field is to choose first an *area* which should contain the necessary petroliferous source beds. The next general step is to obtain the required permission to trespass the land and conduct the investigations. The locale is kept sufficiently large to allow data to be obtained on a regional scale. As the broader subsurface picture develops, the area of economic importance becomes defined. Local property boundaries are not considered too dogmatically until after the subsurface picture begins to take form.

In mining exploration, however, we usually find a different type of thinking guiding the exploration program. It is true that the mining property may be located in an area where conditions are favorable for ore occurrence. From that point on, however, the exploration attention far too often is rigidly focused within the narrow limitations of specific claim boundaries. The chief criticism of mineral exploration today, from the geophysicist's point of view, is that a sufficiently broad overall picture is not obtained before concentrating on detailed exploration.

Application of Geophysics to Mining Exploration

The application of geophysics to mining exploration may be divided into two general categories: first, its application to areas where detailed studies are dictated, and second, its application to areas where reconnaissance studies may be employed to delineate the zones of interest.

To date, the largest application of mining geophysics has been the detailed type of survey, usually applied to operating or previously worked properties. In such areas, the property lines and claim corners far too often constitute rigid boundaries of the area to be examined. The problem handed to the geophysicist then becomes one of locating an ore body within the confines of that particular property.

The application of geophysical work under these conditions usually is a distress measure. The geophysical work is not being done as part of an intelligently planned exploration program. Far too often, it is resorted to as a last attempt to stave off shut-downs, or to aid in future promotion and financing. When geophysics is employed in this way, it has very little chance of achieving success.

It will be most difficult to fix a reliable statistical figure showing the chance for success of any exploratory program seeking to locate new ore in working or previously worked areas. However, when we consider the high percentage of prospects that are abandoned each year, we know that the chance of locating such ore bodies is usually quite remote.

Even though the chance of finding new ore bodies or extensions of known bodies may be discouraging, geophysics can serve a most useful, albeit negative, function in such mining operations. The history of mining and oil exploration clearly indicates that the greatest single factor causing useless loss of capital has been the common human trait of not knowing when to stop. Human nature being what it is, every attempt is made to prolong the life of a mining venture and to operate the property in hopes that the operation eventually can be profitable. Oftentimes, geophysics can serve as a means of evaluation and can be of definite value in preventing the useless expenditure of the remaining capital assets.

An examination of this type may be illustrated by a geophysical survey of a mine in central California, located in a complexly faulted area of paleozoic sediments (comprising limestones, dolomitic limestones, quartzites, and shales), intruded by numerous dikes and sills, (monzonite porphyry, diabase, and quartz-diorite porphyry). These formations are cut by numerous post- and pre-mineral faults, some of which may have cut off or displaced the ore bodies. The complicated subsurface conditions make it impossible to locate any faulted extensions of the previously mined ore bodies, based upon geological deduction alone. The rocks in that area have densities vary-

ing from 2.2 to 2.8, while the massive galena ore has a density of 7.8, and the oxidized ore a density of 5.2. The surface topography is rugged, but of a configuration amenable to reliable topographic correction. Because of the difference in density between the country rocks and the ore, a gravity survey offered the most expedient and economical means of ascertaining the information desired.

In consultation with the mining engineer of this company, a decision was reached regarding the minimum size of ore body to be considered commercially minable. Calculations were then made to determine the gravimetric effects of such an ore body, at the probable depths, as indicated by the faults encountered in the present underground workings. These calculations showed that such an ore body, if present, would exert a local gravimetric influence in excess of ten times the probable error in the work, and would therefore be readily detectable. Furthermore, the extensive underground workings would allow measurements to be made much closer to the probable places where such faulted ore bodies might exist.

The results of the geophysical work showed that there were no gravimetric anomalies of magnitude even approaching that to be expected from a body of the size under consideration. It was concluded therefore that no such ore body existed. From the viewpoint of finding new ore, the work was unsuccessful. However, from the viewpoint of operation planning and evaluation, the work was most advantageous, because it gave a good answer to the question of "when to stop."

Reconnaissance Geophysical Work in New Areas

I would now like to discuss the application of geophysics in reconnaissance exploration. Here again we can try to evaluate the possibilities of success by studying the history of mining exploration. Estimates by numerous mining engineers indicate that the average number of prospects examined will run from 1000 to 5000 before one property suitable for commercial operation is found.

With such great odds against locating an ore body, it is clearly apparent that the exploration program must be directed primarily toward screening an area, and at the lowest cost per acre. In many cases this can be accomplished by the following steps: (1) geological studies, with reference to structure, intrusive igneous bodies, and mineralogical guides in the area; (2) reconnaissance geophysics over the geologically favorable areas; then (3) detailed geophysical examination of the more favorable zones, to locate anomalies that might be caused by commercial mineralization; followed by (4) direct exploration, such as trenching, shaft, tunnel, or drill hole, to evaluate the anomalous condition.

The use of this technique may be illustrated by a survey in Inyo County, Calif.[4] Previous history showed that a lead-zinc sulphide ore occurred in the limestone, adjacent to a limestone-monzonite contact. This contact initially was mapped at four outcrops over a distance of about a mile. Between outcrops the contact was covered by fill material. This type of sulphide is a good electrical conductor and its presence may be indicated by determining the anomalies or zones of better electrical conductivity. Reconnaissance measurements were made along the indicated line of contact, using an inductive method. This was followed by detailed direct current measurements to map the zones of better electrical conductivity. Two main zones of higher conductivity were mapped, one near an old tunnel. A crosscut to the contact encountered ore, consisting chiefly of "lead sulphide replacements, and the large oxidized ore bodies adjacent to them, which now have been mined continuously since 1940."* Later, the development work was extended to cut the second conductive zone, which was at a depth of about 300 ft below the surface. This zone proved to be high-grade sulphides, nearly underlying the old oxidized ore body previously mined and abandoned.

A good example of a regional reconnaissance survey involves the use of magnetic work which led to the westward extension of the Rand in South Africa.[5] Magnetometers were used to map the sub-outcrop of a metamorphosed ferruginous shale formation buried under approximately 2000 ft of unconformable dolomite. The buried formation is comprised of vertically bedded quartzites, slates, shales, some volcanics, and the gold-bearing conglomerates. The slates are magnetic, and their stratigraphic relation to the main gold-bearing reef is well established. Mapping the magnetic anomaly associated with the slate, therefore, served to locate the desired reef which carries the gold values. As a result of this work, it is estimated that the potential gold reserves of the Rand have been increased about 15 pct.

Another example of regional, geologically-guided geophysical work is that conducted by the International and the Falconbridge Nickel Companies in the Sudbury District, Ontario, Canada.[6] The ore is massive nickel and copper sulphides, with sufficient pyrrhotite to make it magnetic. The surface covering is chiefly glacial out-wash, at depths varying from 50 to 300 ft. After completion of the reconnaissance work, electrical methods were used to check the important magnetic anomalies. The subsequent exploration results have established the success of this type of exploration technique as applied to that area.

Geophysical Work Versus Direct Exploration

Usually, geologically-guided geophysical work may be done more economically than geologically-guided direct exploration. Quite often a comparison is made between the cost of a geological examination and the geophysical examination of a mining property. This comparison is not too enlightening, because geological work alone in the usual complex mining area is most limited in its ability to predict the subsurface conditions. Dips and strikes of exposed rocks must be extended underground with extreme caution, in a folded and faulted area. As the experienced mining engineer and geologist also knows, geological prediction is greatly handicapped until enough development has been done to establish the local habits of ore genesis and occurrence. Geological work then must be supplemented with direct exploration, such as drilling. The cost of such geological work and the necessary direct exploration usually is far greater in expenditure of both time and money, than geologically-guided geophysical work.

Whether geologically-guided direct exploration or geologically-guided geophysical reconnaissance is the best procedure will depend upon local conditions and the information desired. For instance, the cost of a single 300 ft drill hole is equivalent to the cost of geophysical work over many acres.

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* Personal communication from H. E. Olund, U. S. Bureau of Mines, to S. E. Stein, Oct. 4, 1948.

The drill hole should give detailed and specific information of the conditions along the exact path traversed by the drill. On the other hand, the geophysical work should give general information over a relatively large area. Obviously, there can be no fixed rule as to which is the more expedient technique. Generally, however, the geologically-guided geophysical work will be the more economical during the reconnaissance stage, while obtaining the overall picture of the area. Then, the direct type of exploration may be employed to determine the economic value of the geophysical anomalies.

Depth of Geophysical Work

The depth to which reliable geophysical work may be conducted in the exploration for minerals is governed almost entirely by the size and configuration of the subsurface feature, and the physically measurable differences in situ between that feature and the country rocks. The choice of a geophysical method is also important. For instance, a vertical plug or ore deposit about 60 ft across and 200 ft high, having a density about three times that of the surrounding medium, could produce a detectable gravitational anomaly (in an area where the necessary corrections may be made for topography and subsurface changes in formation) at depths of about 700 ft. On the other hand, this same quantity of sulphide ore, if extended and existing as a stringer of mineralization along a vein, might not be detectable by gravity work to a depth of even a few tens of feet. However, such a long extended ore body could easily be detected by the electrical methods, which should be able to show its presence to depths of perhaps 100 to 300 ft.

Even with a depth limitation of one hundred or a few hundred feet at best, there are very large areas where mining geophysics may be useful. In Canada, over 90 pct of the pre-Cambrian shield is covered with shallow water and overburden. In the United States and Mexico there also are many thousands of square miles covered with alluvial fill, lake beds, volcanic flaws, etc., underneath which commercial mineralization probably exists. The Basin and Range Province of the western United States is an outstanding example of this condition.

Our surface areas have been prospected quite thoroughly and the mines of the future will come chiefly from the mineralized zones hidden from surface prospecting today.

Type of Equipment

The complexity of geological conditions which exist in a majority of mining areas is in contrast to the relative simplicity of structure governing the accumulation of petroleum. Discovery of these simple petroleum structures with their large size is usually accomplished with a single geophysical method. In mining exploration, however, we are dealing with relatively small features embedded within a complex environment, and this usually indicates the desirability of using more than one geophysical method. Perhaps this can be explained by considering the familiar analogy of looking at a building from only one side. A single view from any one angle differs from the view obtained from another angle. Two views taken from different angles yield information regarding the size and type of building far in excess of that furnished by either of the single views alone. Knowing the type of ore and its occurrence, the geophysical methods should be chosen to give this complementary type of data.

For instance, the successful use of the magnetometer in reconnaissance exploration for the nickel-bearing ores in Canada is predicated upon the fact that pyrrhotite, a magnetic material, occurs with the nickel. There are, of course, many other conditions that could cause a magnetic high in that area, such as disseminated magnetite or intrusive basic rocks. To differentiate between the sulphide ore desired and some other magnetic condition, a traverse or two with one of the electrical methods would give the necessary complementary information. The self-potential method, when ground water conditions are correct, will detect sulphide bodies, such as pyrrhotite, nickel, etc. If, therefore, under these particular conditions, a self-potential survey discloses an active electrical negative center over the same general area where the magnetic high was mapped, we can be fairly certain that this dual condition is caused by the pyrrhotite in the ore. Drilling or other exploration would then be recommended. If, however, an active electrical center is not obtained over the magnetic high, the chances are that our magnetic work has merely mapped a basic igneous rock or a concentration of magnetite.

Quite often, two types of geophysical measurement may be made during the same survey without an appreciable increase in cost. For instance, magnetic and gravity measurements can be made at each station with an increased time for reading which adds only about 20 pct to the total cost.

Complementary measurements allow more reliable interpretation, and in many cases actually result in a decrease in overall cost of the work by decreasing the detail which would be necessary if only one method were applied.

Interpretation of Data

Interpretation no doubt contains the greatest personal equation of any of the components of mining geophysics. This is due to the human inability of the interpreter to evaluate accurately the full significance of the geological and geophysical data, in terms of economic geology. Interpretation must be guided by experience. The subsurface is an extremely complex, three-dimensional region which the successful interpreter must diagnose with only the help of the far too often limited and incomplete data obtained from surface geological and geophysical work. For this reason, close teamwork of the mining engineer, or geologist, with the geophysicist is an important essential in proper planning of the work, and the interpretation of the data. There can be no substitute for this cooperation between the mining or geological and geophysical personnel.

Geophysical Methods for Mining Exploration

Because of the complexity of subsurface conditions, the geophysical crew must be equipped with the necessary apparatus to allow those different techniques to be applied which will give the most useful information for the least expenditure of time and money. Over a period of years, experience has shown that a properly organized geophysical crew will be able to cope with the problems encountered in the usual mining work, when equipped with the following methods:

1. Electrical: for determining the areas of better electrical conductivity, such as caused by sulphide ore bodies of an elongated configuration. The elec-

trical method must be capable of making rapid lateral measurements for reconnaissance, as well as detailed vertical measurements, and must be of a type which clearly differentiates the deeper, subsurface effects from the masking effects of near-surface inhomogeneities.

2. Self-potential: for determining negative centers, such as created by sulphide ore bodies undergoing oxidation.

3. Gravity: for determining areas of different gravitational pull, as created by massive sulphide or oxide ore bodies, buried contacts, or intrusives.

4. Magnetic: for determining areas of different magnetic susceptibility, such as caused by ores containing pyrrhotite, magnetite, etc., and the basic rocks, and for mapping buried contacts between sedimentaries and basic rocks, intrusives, and flaws.

5. Seismic: for measuring the relative velocities of the materials comprising the subsurface; for obtaining the thickness of fill materials and depth to bed rock, etc. The equipment should be capable of doing shallow work by the reflection or the refraction techniques.

6. Equipment for electrical logging of drill holes.

Geophysics in Drill Holes

Although diamond and core drills are not geophysical equipment, their use is so closely related to the use of geophysics that special mention should be made of equipment for electrically logging drill holes. The function of the geophysical work is to show where subsurface anomalies exist. When drilling rigs are available, they are one means of investigating these anomalies and of obtaining the most direct type of data at the minimum cost. Since drilling is often conducted contemporaneously with the geophysical work, it is advisable that the geophysical equipment include a portable electrical logging outfit. Many stories are prevalent in mining folk-lore, describing ore bodies that have been missed a few inches by the drill hole. Its use in mining exploration undoubtedly will prove to be as valuable as it is in petroleum exploration.

Success Versus Accuracy

It should be borne in mind that very seldom are subsurface conditions so simple that the geophysical measurements will serve as a direct means for locating ore. Under favorable conditions, the geophysical work will locate anomalies or zones that could be ore or perhaps any one or more of a dozen other conditions. Direct exploration must therefore be employed to determine the exact cause of this anomaly. The chief function of the geophysical work is to locate the anomalies.

In petroleum exploration, as has already been stated, geophysical work has a success record of about one producer to three dry holes. The conditions in mining are of greater complexity. Therefore, we must contemplate drilling many geophysical anomalies for each ore body discovered. The success record of mining geophysics should be judged upon its ability to locate these anomalous conditions where commercial ore may exist. Direct exploration will then evaluate the commercial possibilities of the geophysical anomaly.

The officials responsible for exploration planning in the mining industry should give more study to the possible role of geophysics in their exploration programs. They should then adjust their budgets so that geophysical work may be coordinated with the initial exploration planning. By so doing, geophysics can become part of the regional studies and be best utilized as a tool for exploration.

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CORRECTION FEBRUARY ISSUE

"The Flotation of Copper Silicate from Silica" By R. W. Ludt and C. C. DeWitt.

On page 49, column 3, the formula shown is the formula of the leuco-base of the dye. The general formula should have been reported as follows:

$$
\begin{array}{ll}
 & C_6H_4{\cdot}N(CH_3)_2 \\
 & \quad \cdot \\
p\text{—}R\text{—}C_6H_4{\cdot}C & \\
 & \quad \ddot{} \\
 & C_6H_4{:}N(CH_3)_2 \\
 & \qquad \cdot \\
 & \qquad Cl
\end{array}
$$

Cyclone Operating Factors and Capacities on Coal and Refuse Slurries

By D. A. DAHLSTROM,* Junior Member AIME

Introduction

Although the liquid-solid cyclone is a relatively recent innovation in the field of coal preparation, various authors have already indicated three distinct applications to operations encountered in the modern tipple. Driessen's articles and work at the Dutch State Mines exhibited its possibilities in the beneficiation of fine coal with apparently high efficiencies down to the 200-mesh fraction.[1,2,3,4] Other publications point out its use as a recoverer, thickener, and preliminary dewatering agent of fine coal which in too many cases is being wasted due largely to former high operating cost limitations.[5,6] Finally, certain authors have emphasized the practicability of applying the cyclone to the elimination of coarse and fine solids from water slurries in the tipple in order to produce a water suitable for recycle purposes containing only minute particles far below the 200-mesh size.[6] This would result in a closed water system for many cases, with its accompanying economic and operating advantages to locations where water quantity and quality problems are severe; better operation of washing equipment using a consistent water devoid of injurious fine particles; lowered maintenance costs due to the lesser abrasion resulting when the fine abraiding particles are removed; and the greater ease of disposing of refuse streams of reduced volume containing greater percentages of solids.

While the three applications have been indicated to be feasible, only a meager amount of data is available on capacities and solid elimination efficiency as a function of design variables and cyclone friction loss. This information is essential for the optimum use and construction of the solid-liquid cyclone, and accordingly, work on the problem was initiated at Northwestern University.

For the unfamiliar reader, briefly the cyclone consists of a cylindrical section mounted above a truncated cone. The feed nozzle enters the cylindrical ring tangentially with the underflow nozzle permitting discharge of the concentrated solids located at the apex of the cone. The overflow nozzle through which the clarified water exits is centered in the cylindrical section at the top of the cyclone. Half sections of experimental cyclones will be found in Fig 1*a* and *b*. The feed slurry enters with a tangential velocity, thus creating a spiral pattern of high centrifugal force. The solid particles of sufficient size and gravity are ejected outward to the walls and spirally discharge to the underflow. Most of the water with uneliminated fine solids moves radially inward along the path of the outer

San Francisco Meeting, February 1949.

TP 2633 F. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Oct. 30, 1949. Manuscript received Feb. 3, 1949.

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[1] References are at the end of the paper.

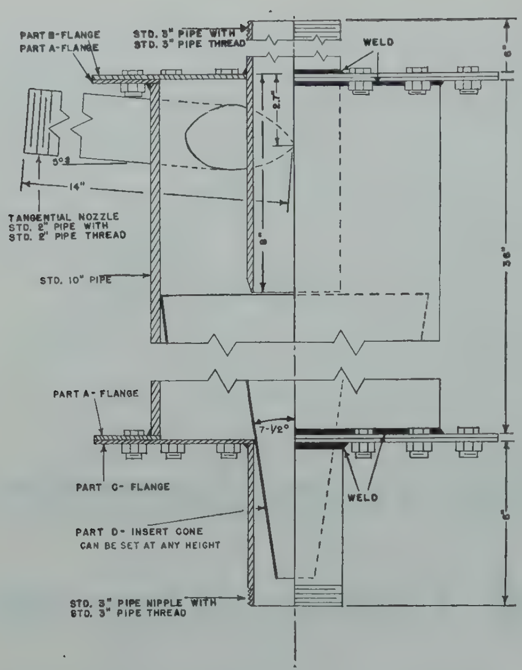

FIG 1*a*—Ten inch Wilmington cyclone, Northern Illinois Coal Corp., Wilmington, Ill.

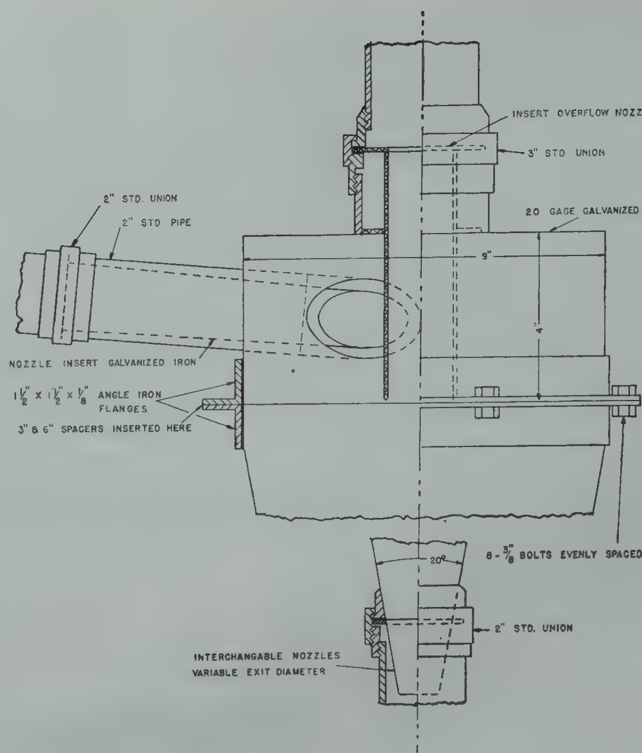

FIG 1*b*—Half section of 9 in. experimental cyclone.

spiral to a second inner spiral at the cyclone core to pass out the overflow. The latter spiral is the most critical fluid movement because of its small radius and higher tangential velocity. The centrifugal force which furnishes the elimination power will attain a maximum at this point.

To the operator, the factors of pressure drop, capacity, and solid elimination efficiency of the cyclone are of paramount importance. Naturally, if he is to design a cyclone for a particular situation, he must know what variables effect these factors and to what extent. From previous data and theoretical studies the following variables were considered to be of major significance in effecting one or more of the three factors:[6] (1) inlet nozzle diameter, (2) overflow nozzle diameter, (3) rate of cyclone throughput, (4) weight percentage of solids in feed slurry, (5) specific gravity of solid material, (6) separation between overflow and conical section, (7) included angle of the conical section, (8) percentage of total volume reporting to the underflow, (9) diameter of cyclone, and (10) type of underflow discharge.

If the effects of these variables can be definitely determined, the economic and operating balance to predict the optimum cyclone design for any application could be performed.

Equipment

In attacking the problem, it was decided that theoretical studies alone would be insufficient because results from actual tipple installations would be necessary for correlating purposes. Therefore, a cyclone utilizing a short length of standard 10 in. casing with removable sheet metal insert cones was constructed at Northwestern University and installed at the Wilmington, Ill., tipple of the Northern Illinois Coal Corp. processing recycle water. Fig 1*a* is a half section of this cyclone and indicates the flexibility of the conical section with respect to position and included angle. During the course of the test runs, the feed nozzle was changed from a 2 in. to a 2½ in. standard pipe.

For further industrial data, permission was obtained from the Truax-Traer Coal Co. to make test runs on their 14 in. Driessen cone, dewatering a fine coal slurry at the Kayford, W. Va., tipple.

The principal cyclone used for the theoretical studies is illustrated in half section in Fig 1*b*. By use of this construction, all the three nozzles, included angle, and separation between overflow and conical section, could be varied. Thus, of the ten important variables affecting operation, only one, cyclone diameter, could not be observed with this equipment. However, this study was made through the use of a 7 in. cyclone used and described in an earlier paper, and a newly constructed 3 in. cyclone. Both of these cyclones were of fixed dimensions and possessed included angles of 20°.

The equipment setup for the theoretical studies is illustrated in Fig 2. When running water only, the water was pumped from the sump tank by two staged centrifugals through a regulating gate valve to the cyclone. The pressure was measured at the inlet to the feed nozzle while overflow and underflow pipes discharged back to the sump tank. During a run, rate samples were taken from the overflow and underflow in weighing tanks for timed intervals, the feed being reconstructed from these data. The same procedure was used for solid runs except that a high-speed mixer was placed in the sump tank in order to maintain a consistent slurry concentration. In addition, small samples of overflow and underflow were taken for analysis work.

Pressure Drop Studies

For convenience in study, the problem was divided into two parts, cyclone pressure drop or energy loss, and solid

elimination efficiency. The two divisions will be correlated at the termination of the paper.

In considering pressure drop, it was necessary to refer to the gas-solid cyclone theory as a starting point. According to Shepherd and Lapelle, the capacity equation for the gas-solid cyclone is a function of inlet and overflow diameters and is expressed in terms of cyclone energy loss divided by the inlet velocity head as follows:[7,8,9]

$$F' = K' \frac{b^m}{e^n} \qquad 1]^*$$

As this equation is difficult to handle and usually involves a trial and error solution, the following transformation was made.

By definition, $F' = \dfrac{F}{v^2/2g} = K' \dfrac{b^m}{e^n}$

where F is the energy loss expressed as feet of fluid.

Rearranging and taking the square root gives

$$\frac{v_i}{\sqrt{F}} = \sqrt{\frac{2g}{K'}}\,\frac{e^{n/2}}{b^{m/2}} \qquad [2]$$

However, the gallons per minute of feed slurry is equal to

$$Q = \frac{60\pi b^2}{4}\left(\frac{1}{231}\text{ cu in. per gal}\right)12v_i \qquad [3]$$

or

$$v_i = 0.408\,\frac{Q}{b^2} \qquad [4]$$

Substituting for v_i in Eq 2 and rearranging, gives

$$\frac{Q}{\sqrt{F}} = \frac{19.7}{\sqrt{K'}}\,\frac{e^{n/2}}{b^{m/2-2}} \qquad [5]$$

This equation can be further simplified by combining all constants and exponents to one symbol, as follows:

$$\frac{Q}{\sqrt{F}} = K(e)^s(b)^t \qquad [6]$$

where $K = 19.7/\sqrt{K'}$
$s = n/2$
$t = 2 - m/_2$

The term, $Q/\sqrt{F}$, has been designated as the capacity ratio and is the basic expression for any flow apparatus, i.e., capacity is a function of the square root of energy loss, and thus $Q/\sqrt{F}$ should remain constant for any constant dimension apparatus. It will be observed that Eq 6 is expressed in simple terms, easily measured, and involving no trial and error in its use. Thus, if valid, it represents a rapid and easy way of predicting either capacity or pressure requirements for any cyclone. It should be emphasized at this time that the term F represents the energy loss in the cyclone proper. In the complete installation, sufficient energy will have to be supplied to overcome all friction losses and potential and kinetic energy changes in the feed line and nozzle and overflow pipe and nozzle.

Taking the logarithm of both sides of Eq 6,

$$\log (Q/\sqrt{F}) = \log K + s \log e + t \log b \qquad [7]$$

As K, s and t have been assumed as constants in the above derivation, Eq 7 represents a straight line if either e or b are held constant. Therefore, if the theory advanced is valid, a plot of capacity ratio as a function of one diameter with parameters of the other should yield straight line curves on log-log paper. Furthermore, all parameters of one set should be parallel with a slope of either s or t. The K value can be determined from the intercept value for capacity ratio where the variable diameter is equal to 1 in. by use of Eq 7.

To test the theory's validity, water runs were made with the 9 in. cyclone varying the inlet and overflow diameters. By collecting rate samples of underflow and overflow, friction losses occurring before the cyclone inlet and after the overflow could be calculated. Finally, by applying Bernoulli's flow equation between the point of pressure measurement and the overflow exit pipe, the cyclone energy loss was determined. This method neglects the energy remaining in the underflow discharge stream. As this energy is largely present as rotational kinetic energy with the vortex type of discharge, it is difficult to measure. In order that this error would be small, underflow discharges were generally held to less than 5 pct by volume. However, it should be pointed out that to maintain the vortex underflow discharge, this energy could not be recovered in industrial applications and therefore might better be considered in the energy loss of the cyclone.

Capacity ratio expressed as feed gallons per minute divided by the square root of the cyclone energy loss in feet of fluid determined for the various runs was plotted against inlet diameter with parameters of overflow diameter in Fig 3*a*, and against overflow diameter with inlet diameter parameters in Fig 3*b*. Both are log-log plots. It will be observed that good straight parameter lines could be drawn for both cases, all of which were parallel for any one graph. Furthermore, this straight line relationship is valid for a ratio of overflow to inlet diameter of 0.6 to 2.0. This ratio more than covers the usual industrial cyclone design.

To correlate these findings with industrial data, test runs on the 10 in. Wilmington cyclone are included in Fig 3*a* and the 14 in. Kayford Driessen cone in Fig 3*b*. Lines parallel to the pure water parameters were drawn

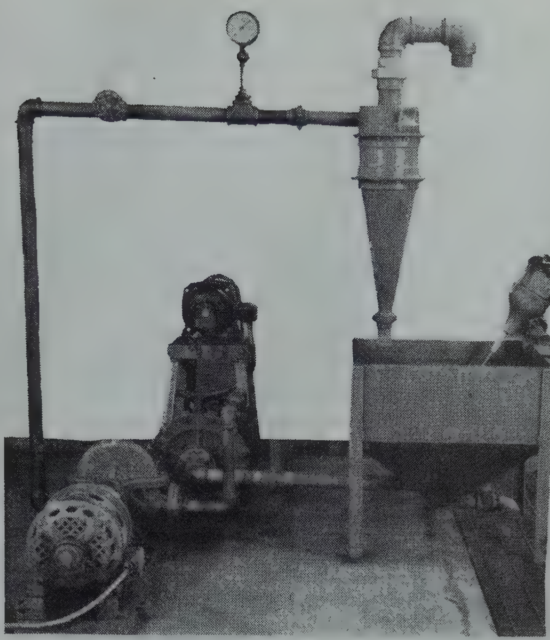

FIG 2—Test unit of 9 in. cyclone.

* Nomenclature is given on p. 344.

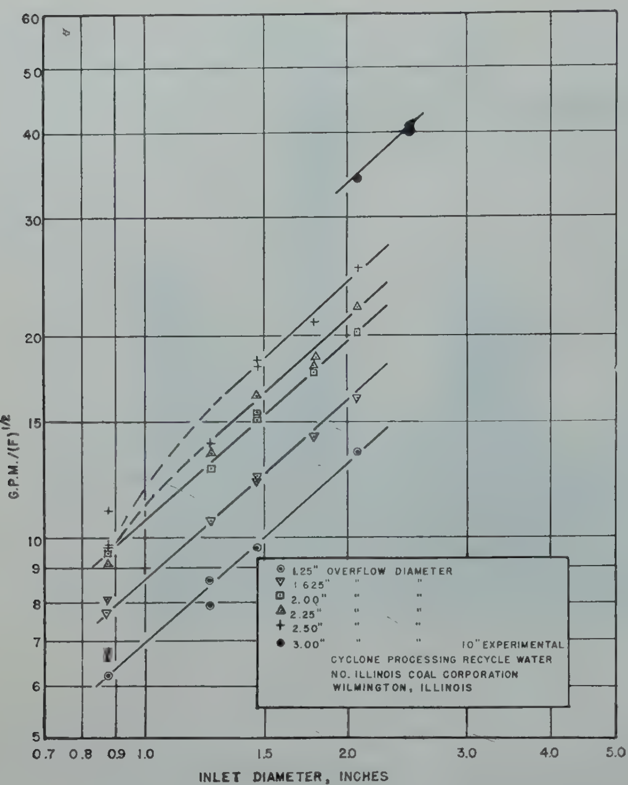

FIG 3*a*—Capacity ratio as a function of cyclone inlet diameter with constant overflow diameter parameter.

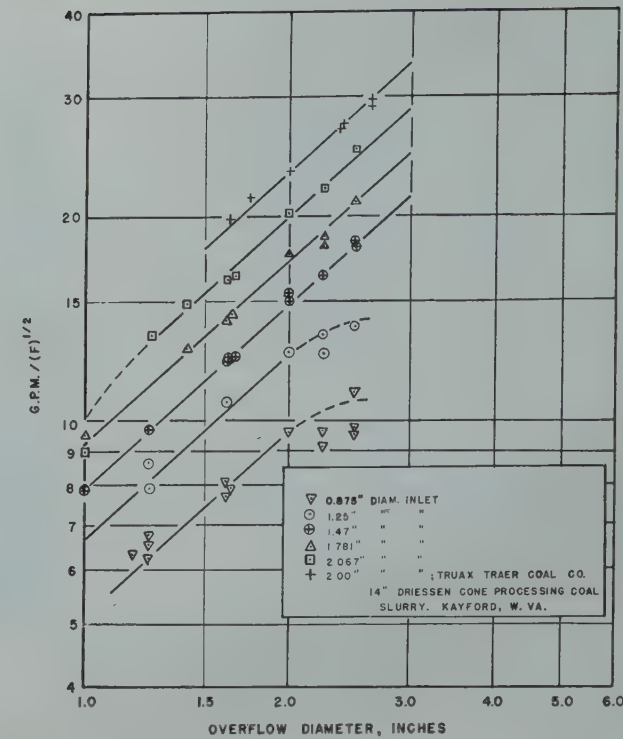

FIG 3*b*—Capacity ratio as a function of cyclone overflow diameter with constant inlet diameter parameter.

through the industrial data to indicate correlation. It will be observed that in both cases agreement was good.

The slope of Fig 3*a* which equals the exponent t is 0.89, while that of Fig 3*b* which equals the exponent s is 0.9. For convenience, the exponent on both terms was assumed as 0.9 which introduces a negligible error. Thus, Eq 6 can properly be applied to any liquid-solid cyclone within the designated overflow to inlet diameter ratio as

$$Q/\sqrt{F} = K(eb)^{0.9} \qquad [8]$$

The average K value for the 9 in. experimental cyclone determined from the straight line portion of the parameters was 5.61. The range of variation was 5.47 to 5.73, with an average deviation of 0.079 or 1.4 pct. For the 9 in. cyclone, then

$$Q/\sqrt{F} = 5.61\ (eb)^{0.9} \qquad [9]$$

At this point, it is interesting to note that the exponents m and n for the gas cyclone were both equal to 2. From Eq 6, therefore, s and t should each be equal to 1 for the gas-solid cyclone. The slight deviation for the liquid-solid cyclone can probably be explained by the different fluid characteristics.

To further illustrate the similarity, the K' value for the gas-solid cyclone was determined as 16.0. From Eq 6, K would therefore equal 4.93. This deviation from the 9 in. K of 5.61 can be attributed largely to minor design factors.

Eq 8 indicates the effect of inlet and overflow diameters and capacity on energy requirements. However, the variation of the K value for any cyclone must be determined, as well as the investigation of the remainder of the 10 variables listed at the beginning of the paper. For convenience, they will be considered individually in the given order.

Weight Percentage of Solids in Feed

Several runs were made with varying solid concentrations in the feed to observe any effect of solid concentration on energy loss. As the line slopes of the industrial data in Fig 3*a* and *b* indicated that solid concentration had no influence on nozzle diameter exponents, any alteration of Eq 9 for the 9 in. cyclone would be found in the K value.

In performing the solid runs, two different types of slurries were used, the first being various coal and refuse solids, and the second a silt and clay material with a close gravity range of 2.70 to 2.75. Assuming Eq 8, K values could be determined for the various runs. However, these values were obviously too high, which seemed incongruous, especially as the feed slurry became more concentrated. Upon observation of the character of the underflow during operation, it was observed that the vortex discharge was considerably more vigorous as the dilution of the solids increased. Thus, it was reasoned that the solids discharged in the underflow represented a negligible amount of energy loss, and therefore should be omitted from capacity ratio considerations. Theoretical principles further strengthen this assumption as cyclone energy loss is almost totally confined to loss of rotational kinetic energy in the inner spiral. As most underflow solids never reach the inner spiral, they should thus be excluded from energy calculations. The vortex discharge is maintained as long as the solid concentration is no more than about 55 pct. Accordingly, in calculating the capacity ratio for solid runs, the volume occupied by the underflow solids in a 55 pct slurry was subtracted from the feed rate in gallons per minute. These values were then plotted in Fig 4 as a function of solid feed concentration. It is apparent that the majority of values fell within experimental error of the water K value of 5.61.

In order to prove this method of calculating capacity ratios for solid runs is correct and has no deleterious effect on the diameter exponent of Eq 8, the industrial data on the 14 in. Driessen cone at Kayford was reworked on this basis. Table 1 is a compilation of these results. As the data had originally been plotted in Fig 3*a* based on the feed in gallons per minute, it was necessary to replot the data, using the corrected capacity ratio. This was done on log-log paper in Fig 5, and also includes the uncorrected data. A line of slope 0.9 was drawn for each set of data and it is apparent that the agreement is as good with the corrected method.

From the above discussion, it is evidently safe to assume that solid concentration will have no decreasing effect on the K value indicating that capacity efficiency is maintained.

Specific Gravity of Solids

From Fig 4, it can be observed that particle gravity has a small or negligible effect on K value. Coal and refuse slurries appeared to have slightly higher values than the heavier material in a few cases. However, it is believed that this influence is minor and can be ignored.

Separation Between Overflow and Conical Section

Because of the change in direction of the outer spiral at the base of the conical section, the placement of the overflow exit with reference to this point can be expected to have an effect on the capacity ratio. For the 9 in. cyclone, 3 and 6 in. cylindrical spacers, which could be inserted between the conical section and overflow point, were constructed. Capacity ratio tests with a parameter of constant overflow and one of constant inlet were made and results plotted in Fig 6*a* and *b*. Included in these log-log graphs are the same parameters obtained from Fig 3*a* and *b* for the 9 in. cyclone without spacers. From the 6 in. parameters of Fig 6*a* and *b*, it is apparent that a 6 to 8 pct increase in capacity ratio was obtained by its use. This probably results from the creation of a more gentle direction change at the base of the conical section when it is removed from the overflow point where undoubtedly other flow point where undoubtedly other severe direction changes are occurring. Due to the lack of time, it was impossible to follow this trend further with

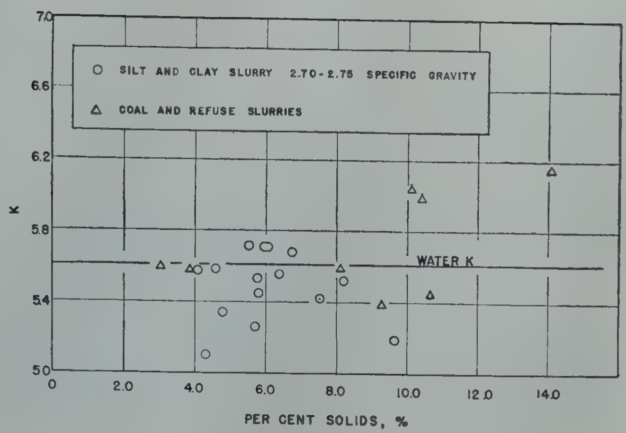

FIG 4—K value as a function of solid concentration, 9 in. cyclone.

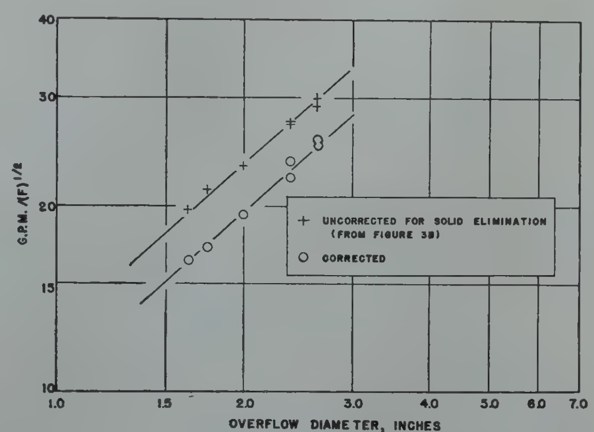

FIG 5—Corrected solid capacity ratio on 14 in. Driessen cone as a function of overflow diameter.

Table 1 . . . Test Runs on a 14 In. Driessen Cone, Truax-Traer Coal Co., Kayford, W. Va., Feed Slurry—Clean Coal Boot Overflow

Run No.	Diameter, In.		Line Pressure, Lb per Sq In. Gauge	Volume Rate, gpm			Volume, Pct To Underflow	Sloids, Pct			Solids Rate, Lb per Min		
	Inlet	Overflow		Feed	Overflow	Underflow		Feed	Overflow	Underflow	Feed	Overflow	Underflow
1	2	2⅝	24.75	212.9	185.1	27.8	13.1	11.42	3.98	56.6	207.5	62.0	145.5
2	2	2⅜	25.25	203.4	174.8	28.6	14.1	11.72	3.98	53.4	204.7	58.5	146.2
3	2	2	26.0	173.6	135.1	38.5	22.2	14.50	4.59	46.0	217.2	52.1	165.1
4	2	1¾	26.25	158.8	109.7	49.1	31.0	15.18	4.40	37.9	209.1	41.0	168.1
5	2	1⅝	25.75	146.5	93.0	53.5	36.5	15.30	4.15	33.3	195.9	32.8	163.1
6	2	2⅝	25.75	222.2	191.0	31.2	14.0	15.12	6.90	58.3	295.0	113.0	182.0
7	2	2⅝	26.5	239.6	236.4	3.2	1.3	14.65	13.81	65.5	305.4	283.2	22.2
8	2	2⅜	26.5	204.2	167.2	37.0	18.1	17.3	7.40	58.5	309.8	106.1	203.7

Run No.	Pct of Total Solids in Underflow	Calculated Cyclone Energy Loss Ft of Feed Slurry	Capacity Ratio		Underflow Discharge Type	50 Pct Point Microns Equiv. Diam	Correlating Factor $(gpm)^{0.53}/(eb)^{0.68}$
			Uncorrected	Corrected			
1	70.2	54.0	29.0	25.2	Transition	47	5.64
2	71.4	55.0	27.4	23.6	Transition	46	5.84
3	76.0	55.6	23.2	19.4	Vortex	41	6.0
4	80.4	55.5	21.3	17.1	Vortex	40	6.25
5	83.3	54.8	19.8	16.0	Vortex	32	6.32
6	61.7	55.3	29.9	25.6	Transition	50	5.69
7	7.3	57.7	31.5	31.1	Overloaded, no hydrometer analysis		5.90
8	65.7	56.8	27.1	22.2	Transition	43	5.84

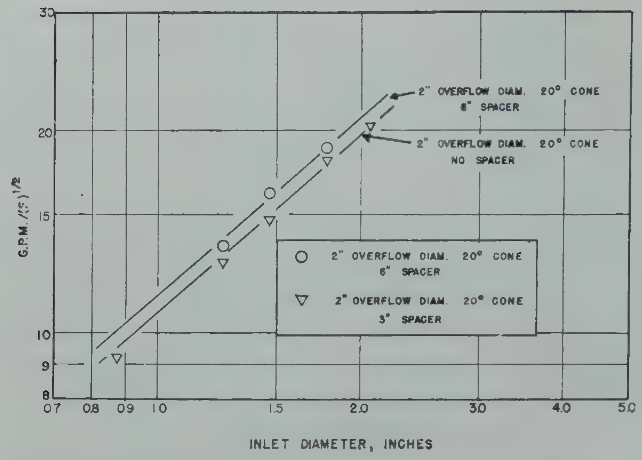

FIG 6*a*—Effect of conical section separation on capacity ratio, 20° cone.

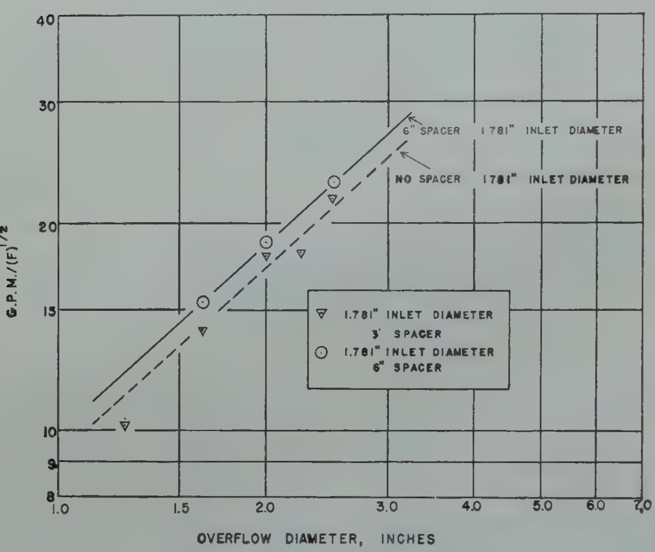

FIG 6*b*—Effect of overflow and conical section separation on capacity ratio.

Table 2 . . . Test Runs, 10 In. Experimental Cyclone, Northern Illinois Coal Corp., Wilmington, Ill., Feed Slurry—Recycle Water

Run No.	Diameter, In.		Line Pressure Lb per Sq. In. Gauge	Volume Rate, gpm			Volume Pct to Underflow	Solids, Pct			Pct of Total Solids in Underflow
	Inlet	Overflow		Feed	Overflow	Underflow		Feed	Overflow	Underflow	
1	2.067	3.067	20.0	229.7	220.5	9.2	4.0	12.5	9.75	59.4	25.4
2	2.067	3.067	19.9	227.3	219.0	8.3	3.7	13.4	11.22	58.2	20.1
3	2.469	3.067	20.75	272.0	260.0	12.0	4.3	12.4	9.7	56.6	25.8
4	2.469	3.067	20.75	272.4	265.5	6.9	2.5	7.6	6.2	49.9	21.2
5	2.469	3.067	22.0	242.8	234.0	8.8	3.6	10.0	7.8	57.6	20.0
6	2.469	3.067	23.4	280.5	268.0	12.5	4.5	8.5	6.31	46.2	29.5
7	2.469	3.067	23.1	295.8	282.0	13.8	4.7	4.3	3.16	25.1	30.4
8	2.469	3.067	21.2	282.2	262.0	22.2	7.15	11.7	8.70	45.4	31.7

Run No.	Calculated Cyclone Energy Loss Ft of Feed Slurry	Capacity Ratio		50 Pct Point Microns Equiv. Diam	Correlating Factor $(gpm)^{0.53}/(eb)^{0.68}$	Cone Included Angle	Overflow and Conical Section Spacing, In.
		Uncorrected	Corrected				
1	45.3	34.1	32.7	23	5.01	15	0
2	44.1	34.2	32.9	Undeterminable	5.00	45	20
3	42.8	41.5	39.7	26	4.94	15	0
4	43.6	41.2	40.2	33	4.94	45	10
5	46.4	35.6	32.9	35	4.66	45	0
6	48.9	40.1	39.0	No hydrometer analysis	5.04	15	0
7	48.4	42.5	41.7	No hydrometer analysis	5.15	15	0
8	44.0	40.7	40.1	29	5.05	15	0

larger spacers, but it is believed that relatively little, if any, increase would have been experienced.

As a corollary of this, data was obtained on the experimental 10 in. cyclone at Wilmington, Ill., a tabulation of which appears in Table 2. A 45° insert cone used during some of the tests could be set at any desired distance from the overflow point. It will be observed that a 16 pct increase in capacity ratio and *K* value was obtained when this distance was changed from 0 to 10 in. No further increase was noted with a larger separation.

Cyclone Included Angle

Due to the more severe direction change, it was believed that capacity ratio and *K* value would decrease with an increase in included angle. A 30° cone was constructed for the 9 in. cyclone and capacity ratio tests performed at a constant inlet diameter, and also one at constant overflow. Results are plotted on log-log paper in Fig 7*a* and *b* and include the same parameter using the 20° cone with no spacer obtained from Fig 3*a* and *b*. A 10 pct decrease in capacity ratio was experienced with the 30° cone with no spacer as compared to the 20° cone. However, values increased as before as spacers were inserted until they slightly exceeded those of the 20° cone with no spacer.

This phenomenon can also be observed in Table 2 where the 15° cone exhibited a 16 pct larger capacity ratio than the 45° cone with no separation between overflow and conical section. However, when the spacing was sufficient, capacity ratios were equal. This also yields the important indication that a 15° cone appears to give the maximum capacity ratio, as the 45° cone could never exceed this value regardless of the separation between overflow and conical section, other dimensions being equal.

Volume Percentage Reporting to the Underflow

Depending on the solid concentration, different volume percentages of the original feed may be discharged to the underflow. To determine this effect, several water runs were made with varying volume distributions between overflow and underflow. The calculated capacity ratios were then compared with those of similar parameters

in Fig 3*a* and *b*. A summary of these tests, together with the calculated capacity ratios and their percentage of deviation from the low volume split ratios will be found in Table 3.

From the percentage of deviation values given in Table 3, it would appear that volume split has only a slight effect on capacity ratio, the majority of runs showing a very small positive increase. Therefore, it seems reasonable to expect Eq 6 to be valid within 5 pct, regardless of the volume split. This is logical when one considers that the vortex underflow stream issues with a very vigorous spray representing a high rotational kinetic energy. The energy must be entirely lost if the vortex discharge is to be maintained. It appears, therefore, that the energy recovered by allowing a lesser overflow is almost completely offset by the energy leaving as rotational kinetic energy in the increased underflow stream.

Industrial data again tended to show the same phenomenon. From the Table 1 tabulation of runs made on the 14 in. Driessen cone at Kayford, W. Va., it will be seen that volume split to the underflow varied from 14.0 to 36.5 pct for the underflow discharge. Yet, it is apparent from Fig 5 that the capacity ratio line was unaffected.

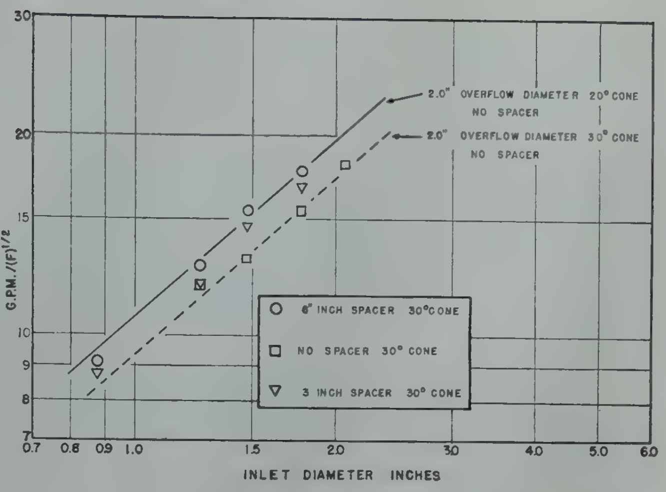

FIG 7*a*—Effect of cyclone included angle on capacity ratio.

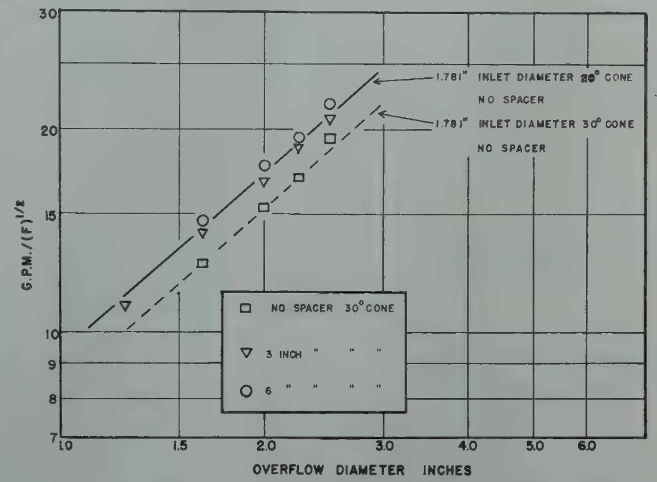

FIG 7*b*—Effect of cyclone included angle on capacity ratio.

Cyclone Diameter

With the gas-solid cyclones, wall friction was found to be negligible, and consequently cyclone diameter had no effect on the *K* value.[7,8] To determine the importance of this variable with the liquid-solid cyclone, a 7 in. cyclone on hand, and a newly constructed 3 in. cyclone were used. Both possessed included angles of 20°, and similar spacings between overflow point and conical section to that of the 9 in. cone without spacers. Water tests on these cyclones yielded the following *K* values, assuming Eq 8 to be valid: 3 in. cyclone, 5.55; 7 in. cyclone, 5.20.

These compare very favorably with the 9 in. cyclone *K* value of 5.61. Solid runs on the smaller cyclones also yielded comparative values. As the 14 in. Driessen cone was similar in construction to the experimental cyclones, its *K* value is also of importance. Referring to Fig 5, the *K* value was determined from the line of the corrected data as 5.57. From these results it can be seen that cyclone diameter actually has a negligible effect on *K* value and therefore an average of the 4 cyclone diameter values would be appropriate. This would then yield the following equation which would be valid for any cyclone of 20° included angle with less than 1 in. separation between overflow and conical section.

$$\frac{\text{gpm}}{\sqrt{F}} = 5.48\ (eb)^{0.9} \qquad [10]$$

It should be emphasized, at this point, that the gallons per minute (gpm) expressed in this equation is the corrected value as explained in the previous discussion on the effect of solid concentration.

The 10 in. cyclone at Wilmington, when used with a 15° cone, accordingly possessed a higher *K* value of 6.38. Thus, the equation for the 15° cone would be

$$\frac{\text{gpm}}{\sqrt{F}} = 6.38\ (eb)^{0.9} \qquad [11]$$

Further interpretations on the application of these equations when used on cyclones of varying design will be

Table 3 . . . Effect of Volume Distribution on Capacity Ratio, 9 In. Cyclone—Water Runs

Run No.	Nozzle Diameter, In.		Volume Pct to Underflow	$\frac{\text{gpm}}{(F)^{1/2}}$	Pct Deviation from Low Volume Split
	Inlet	Overflow			
14	1.25	1.0	28.8	6.93	+4.68
16	1.25	1.406	12.4	9.09	+1.0
19	1.25	1.1875	18.6	7.99	+3.23
26	1.47	1.	26.1	7.86	0
32	2.067	1.406	8.7	13.97	−3.66
94	1.25	1.25	16.7	8.64	+6.53
110	0.875	1.25	17.9	6.22	0
127	1.781	1.25	15.2	10.19	−9.02
131	1.47	1.25	28.1	9.7	0
142	1.47	1.25	27.2	9.02	0
151	0.875	1.25	21.5	6.55	+5.3
176	1.781	1.25	8.6	10.38	−7.73
208	1.781	2.00	8.43	18.0	+4.65
269	1.781	1.625	13.8	14.7	+3.16
271	1.781	2.00	8.8	17.61	+2.38
277	1.781	2.00	8.1	17.7	+2.91
279	1.47	2.00	8.6	15.45	+4.21
281	1.25	2.00	9.1	12.73	+1.84

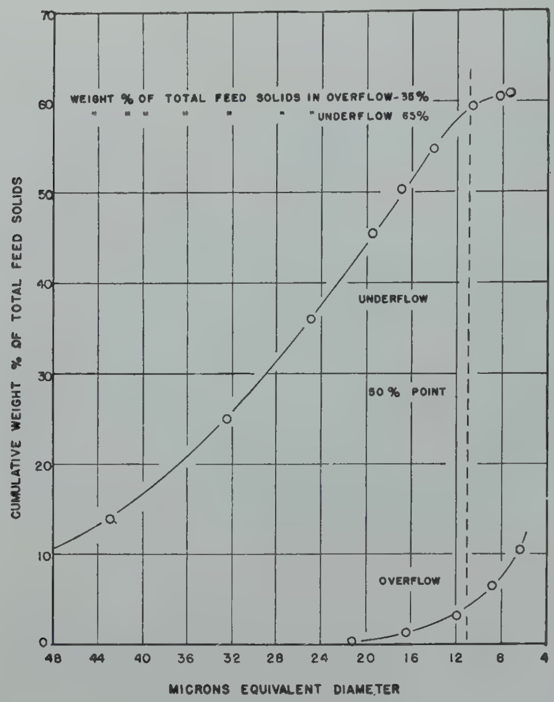

FIG 8—Typical results of hydrometer analysis.

given in the summary at the end of the paper.

Type of Underflow Discharge

Three types of underflow discharge can be obtained on the cyclone. The first is the vortex type which is characterized by a spiral spray with an accompanying air core. The air core is maintained by a vacuum created by the cyclone flow pattern. The second type is commonly designated as overloaded and consists of a very sluggish discharge with no spiral characteristics. Solid concentration is always higher than 60 pct and there is no air entering the cyclone. The condition is brought about by the fact that more ejectable solids are present in the feed than can issue in the underflow. Accordingly, a bed of solids is built up in the apex of the cone and a considerable quantity of solids is carried out in the overflow stream.

No observations of any extent were made with the overloaded condition as all tests to date indicate that solid elimination efficiency is severely injured with this phenomenon in effect. Examples of this inefficiency will be given in more detail in the section on solid elimination. The few solid tests made exhibited an appreciable increase in K value, assuming Eq 8 to hold for the overloaded condition. However, due to the absence of the air core with the overloaded discharge, undoubtedly a different flow pattern is in effect, and therefore Eq 8 should probably be altered.

The third type has been labeled the "transition" discharge as it marks the region between the completely overloaded and complete vortex conditions. It is characterized by a slightly less vigorous vortex than the complete vortex discharge and underflow solid concentration closely approaches that of the overloaded type. Capacity ratio tests performed with this discharge indicated only minor deviations from the established capacity ratio equations. This was correlated by industrial data on the 14 in. Driessen cone.

Yancey and Geer obtained a discharge that appeared to compromise the overloaded and complete vortex conditions.[5] A sausage type of discharge that issued in a rotating fashion was experienced on certain coal slurries with a 5 in. cyclone. No such underflow could be obtained in this work, and possibly the particular size consistency and different cyclone design they were operating with may have brought about the condition.

Besides the big advantage of high solid efficiency, the vortex discharge is much simpler to operate. With the overloaded and transition type, very small changes in feed concentration can bring about a very large loss of solids to the overflow. This is not the case with the vortex type. Illustrations to be given in later discussions will indicate that the very large increase in solid elimination efficiency with the vortex type of discharge considerably more than offsets any small saving in energy requirements with the overloaded discharge.

Solid Elimination Efficiency

In operation, the fraction of solids eliminated from a feed slurry to a liquid-solid cyclone is probably more important than the accompanying energy loss. Operating costs of the cyclone are very small and many times little could be saved by decreasing pressure drop. In contrast, by designing a cyclone to extract a finer particle size there is much to be gained as added revenue in the case of fine coal or decreased operating cost when working with refuse slurries.

The most common method of measuring extraction efficiency is the determination of the 50 pct point. This is represented by the particle size that reports 50 pct to the overflow and 50 pct to the underflow. It also quite accurately portrays the limit of extraction efficiency as very little material of similar specific gravity but coarser in nature will be found in the overflow stream and by the same token only a relatively small amount of particles finer than this dimension will report to the underflow. The phenomenon is pictured in Fig 8, an analysis of a test run on the 9 in. cyclone illustrating the size distribution of the feed solids between overflow and underflow. A vertical line has been drawn at the 50 pct point (location on curves where slopes are equal) to indicate the small amount of material present on one side of this line for either stream.

In previous work, many runs were made with coal and refuse slurries in an attempt to obtain fundamental data.[6] However, this procedure had to be abandoned as the large variation in particle gravity found in this material prohibited the determination of the 50 pct point with any degree of accuracy. Naturally, the elimination efficiency will be a function of solid specific gravity and as solids in these slurries range in gravity from 1.2 to 5.0, it follows that actually a continuous curve of 50 pct point as a function of particle gravity is in effect.

To eliminate this error, it was de-

cided to standardize the tests upon a material that possessed a very close gravity range. This eliminated the use of coal as refuse material of higher gravity would be liberated upon degradation. A silt and clay material of gravity range 2.70 to 2.75 was finally obtained from a local deposit.

Because the 50 pct point for the liquid-solid cyclone is well below the 200 mesh size, screen analyses are of no use in its accurate determination. The hydrometer method of Casagrande, widely used in soil analysis, offered the best solution to this situation.[10] As the analysis is based upon settling velocities, the particle sizes determined are expressed as equivalent diameters—the diameter of a sphere of similar specific gravity settling at the same terminal velocity as the particle in question. Although this may not correspond to the actual particle dimension, it yields a size which depicts the nature of the particle under actual operating conditions. From rate samples taken for each run, it was then possible to determine the 50 pct point. A typical analysis has been plotted on Fig 8 on a considerably reduced scale.

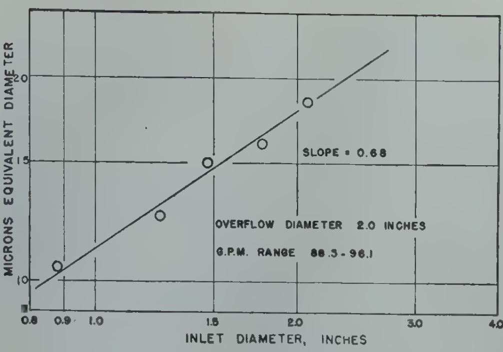

FIG 9*a*—Elimination efficiency as a function of inlet diameter.

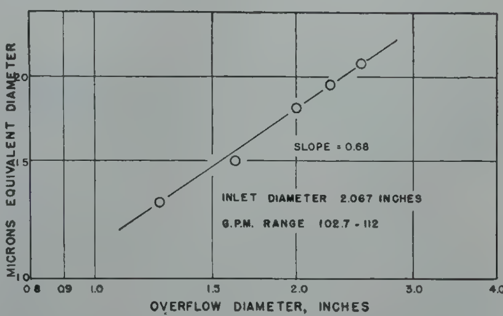

FIG 9*b*—Elimination efficiency as a function of overflow diameter.

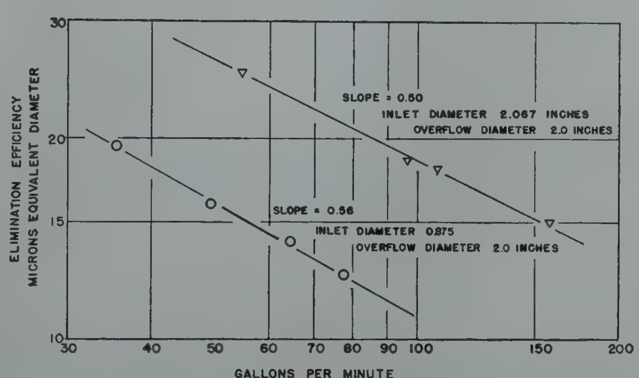

FIG 9*c*—Elimination efficiency as a function of cyclone capacity.

Fundamental Considerations

In attacking the problem, it was felt that the same three variables of major significance with regard to energy loss would be of similar importance in solid elimination efficiency. As inlet nozzle diameter decreases, entrance velocity increases for a constant throughput. Due to the spiral flow pattern, tangential velocity increases inversely with cyclone radius. Therefore it would seem proper to assume an increasing critical centrifugal force as inlet diameter decreased for a fixed capacity. By the same consideration, centrifugal force should also vary directly with volume throughput. Finally, with regard to the overflow nozzle, it is necessary to refer again to the gas-solid cyclone theory.[7,8,11] It was found that the location of maximum centrifugal force occurred at a critical radius approximately equal to that of the overflow tube. By decreasing this radius, centrifugal force should be increased.

To determine the extent of these various factors, elimination efficiency tests were made with the 9 in. cyclone maintaining two of the three variables constant during any one set of runs. The elimination efficiency was plotted against the variable factor on log-log paper to ascertain if the effects were exponential functions. Fig 9*a* indicates elimination efficiency as a function of inlet diameter. During these tests, overflow diameter was 2 in. and the feed rate maintained at 88.5 to 96.1 gpm. Considering that experimental error probably limits determination of the 50 pct point to an accuracy of 1 micron, it appears that the inlet diameter either is or closely approximates an exponential function. The slope of the average line drawn in Fig 9*a* is 0.68 and equals the exponent on the inlet diameter term.

Fig 9*b* plots elimination efficiency vs. overflow diameter during which time the inlet nozzle was 2.067 in. and slurry rate varied between 102.7 and 112.0 gpm. An excellent straight line relationship was obtained with the slope being 0.68 as in Fig 9*a*.

The factor of feed rate is investigated in Fig 9*c* where two sets of runs are plotted against gallons per minute. Straight lines of slopes −0.50 and −0.56

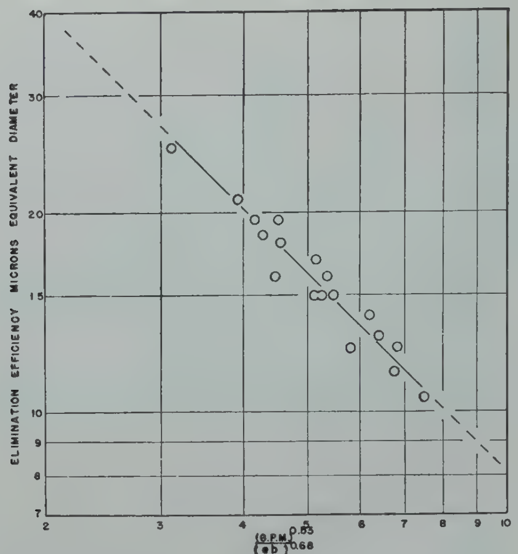

FIG 10—Correlation of elimination efficiency for 9 in. cyclone, 20° cone, no spacers, 2.73 particle specific gravity.

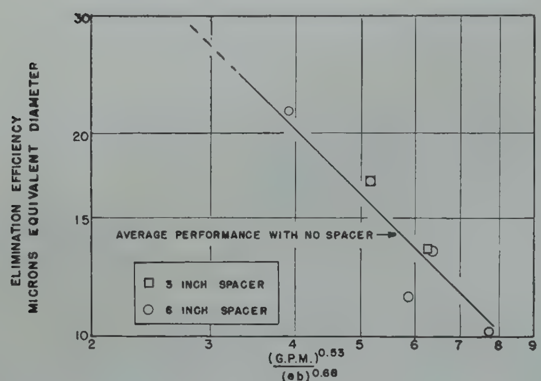

FIG 11—Effect of distance between overflow and conical section on elimination efficiency.

are observed; the slight discrepancy is probably due to experimental error and therefore the average value of -0.53 was used in later work.

From the above results, elimination efficiency must be a function of the three expressed variables raised to the proper exponent. Due to the success of the equation developed for energy loss, a correlation was attempted on the basis of multiplication of the three variables. In order to obtain a small correlating factor greater than one, the reciprocal of the number indicated by the exponents was used. Thus, the correlating factor was:

$$\frac{(gpm)^{0.53}}{(eb)^{0.68}} \qquad [12]$$

Several further runs were made and the elimination efficiencies plotted against the correlating factor on log-log paper in Fig 10. If no other factors are of significant importance exclusive of those which will be discussed in later sections and which were held constant during these runs, it should be possible to draw a line of slope -1 through the results. This was done as indicated in Fig 10 and an observed good agreement does exist. Of the 18 points, only 3 deviate by more than 1 micron and of these, only one is displaced from the line by 2 microns. The equation of the indicated line is:

elimination efficiency, microns

$$= \frac{81\ (eb)^{0.68}}{(gpm)^{0.53}} \qquad [13]$$

It is believed that the range for the 9 in. cyclone has been quite thoroughly explored as gallons per minute varied from 35.5 to 150.6, inlet diameter from 0.875 to 2.067 in., and overflow diameter from 1.25 to 2.50 in.

No similar correlation could be obtained for elimination efficiency with industrial data as indicated in Tables 1 and 2. Two sources are responsible for this situation. First it was impossible to determine the 50 pct point with sufficient accuracy as it appeared to exist over a wide range instead of at an isolated size as with the standard material. This is caused by the wide variation in particle gravity present. Secondly, these samples were taken on many different days and times, undoubtedly large variations in solid consist and quality existed. However, it is obvious from Tables 1 and 2 that elimination efficiency does vary with the correlating factor in the same direction as the test material. Of particular interest is the quantitative efficiency of extraction experienced with the cyclone. Little difficulty was encountered in obtaining a 50 pct point of 15 to 17 microns with a throughput of 150 gpm at 20 to 25 psi pressure. This left in suspension only very fine particles of a nonabrasive character and a water that was suitable for further process work. This represents a capacity of 340 gpm per sq ft area for the 9 in. cyclone. If increased elimination efficiency is desirable, smaller nozzles can be used at higher pressures.

Separation Between Overflow and Conical Section

The distance between the conical section and overflow exerted a definite effect upon the capacity ratio of the cyclone. To check this factor with regard to elimination efficiency, runs were made with the 3 and 6 in. spacers in place. Results have been plotted on log-log paper in Fig 11. Also included is the line drawn in Fig 10 representing the average elimination efficiency performance without spacers. Spacer points were either within 1 micron of the line or well below it in all cases and thus operation can be considered equal to the former method using no spacers. The same phenomenon was observed with industrial data on the 10 in. cyclone. Run 4 of Table 2 which possessed a 10 in. spacing between overflow and conical section exhibits a lower 50 pct point and a higher solid extraction than run 5 where there was no separa-

tion present. This is especially significant as a higher capacity ratio is also obtained with the separation. In future cyclone design, advantage should be taken of this characteristic by allowing 6 to 10 in. between overflow and conical section.

Cyclone Included Angle

Three solid runs were made using the 30° cone to observe the effect of cyclone included angle on elimination efficiency. In these tests, nozzle diameters and capacity were varied in order to yield a good range of the correlating factor. Results are plotted on log-log paper on Fig 12 and are compared with the average performance line obtained with the 20° cone. An appreciable increase in the 50 pct point is readily apparent indicating that included angle should be kept to a minimum.

A similar effect is evident in the industrial data for the 10 in. cyclone of Table 2. Substantial increases in the 50 pct point and corresponding decreases in percentage of total solids eliminated were experienced when the insert cone was changed from 15 to 45°.

Cyclone Diameter

As centrifugal force is inversely proportional to the radius of curvature, investigation of cyclone diameter is mandatory for a thorough examination of elimination efficiency. Solid runs were accordingly performed on 3 and 7 in. cyclones. As the inlet and overflow diameters were fixed it was necessary to vary the volume throughput in order to obtain different values of the correlating factor. Resultant points are indicated on Fig 13 along with the average performance line of the 9 in. cyclone. All values lay within 1 micron of the average line or its extension indicating that cyclone diameter has a negligible effect on elimination efficiency as predicted by the 50 pct point. Furthermore the 3 in. cyclone points always lay above the average line while those of the 7 in. were below, a second indication of the unimportance of cyclone diameter.

Percentage of Feed Volume Discharged to Underflow

Theoretical considerations naturally indicate a decreasing 50 pct point as volume split to the underflow increases. As higher volume fractions report to

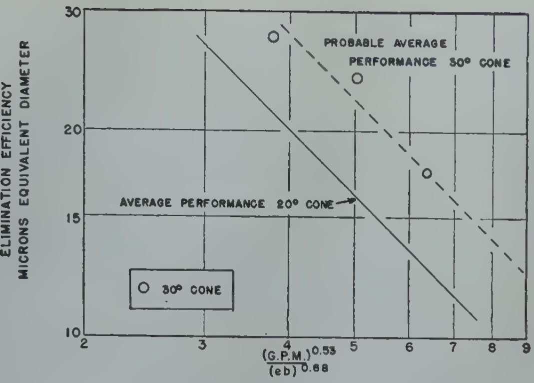

FIG 12—Effect of cyclone included angle on elimination efficiency.

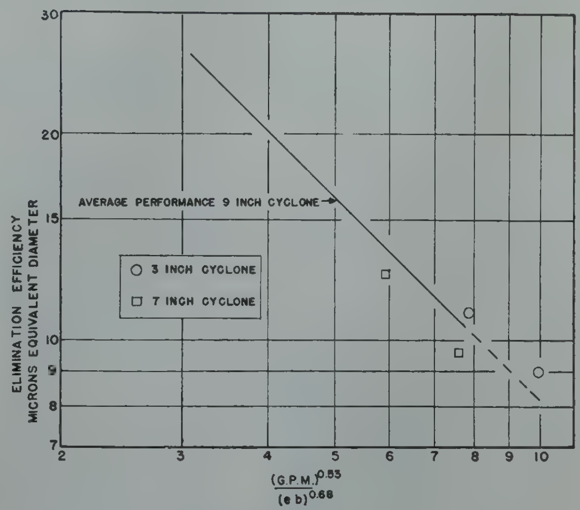

FIG 13—Effect of cyclone diameter on elimination efficiency.

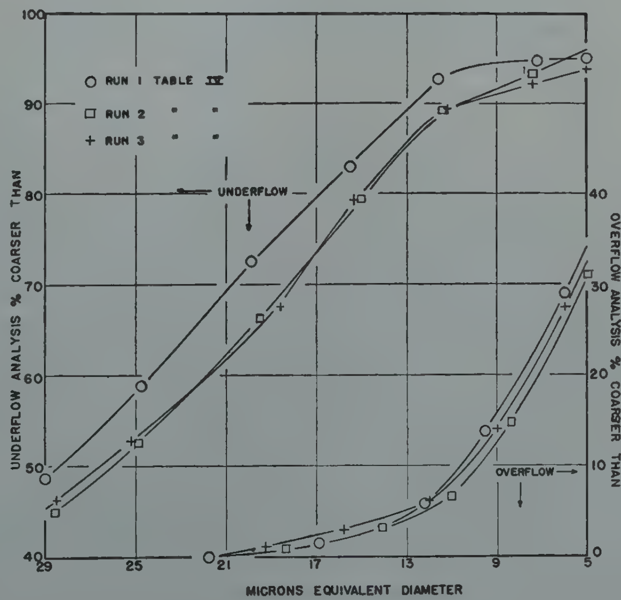

FIG 14—Size analyses of runs 1, 2, and 3 of Table 4.

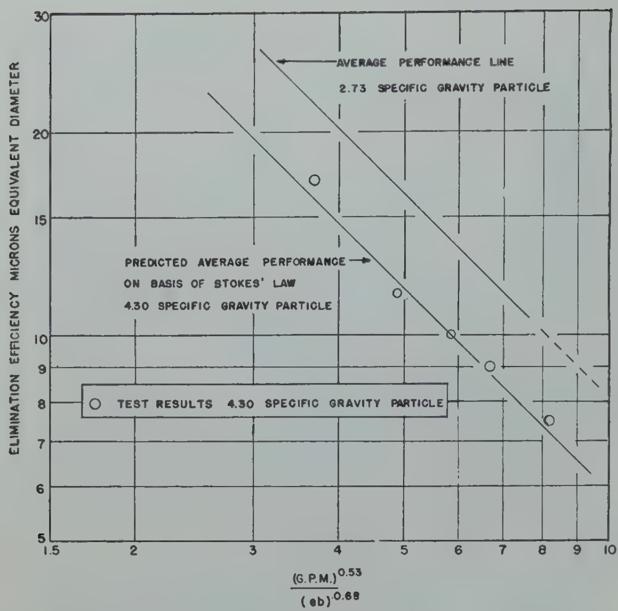

FIG 15—Effect of particle specific gravity on elimination efficiency.

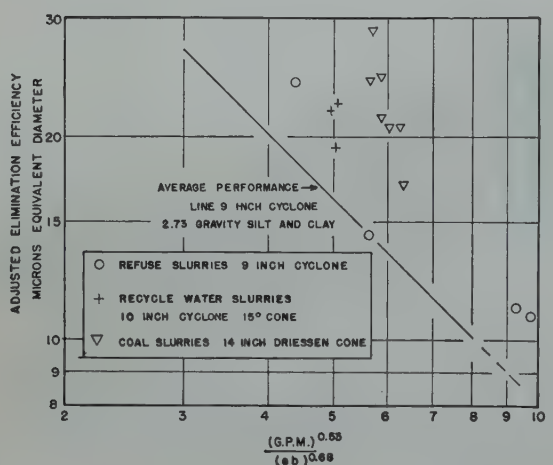

FIG 16—Comparison of coal and refuse slurries solid elimination efficiency with standard 2.73 gravity material.

the underflow, water will be robbed from the overflow and with it accompanying fine solids below the 50 pct point in size. The question that must be answered is whether any further concentrating action takes place by the higher volume split.

Volume split alone is not the only criterion that should be considered. Underflow dilution is also significant as it is a measure of the amount of excess water being directed to the underflow. For coal and refuse slurries, approximately 50 to 55 pct solids seems to be appropriate underflow concentration to maintain a vortex discharge. Any lesser concentration indicates the presence of excess water.

From tests performed on the standard material, no concentrating action seemed apparent through higher volume splits to the underflow. In the correlation of elimination efficiency for the 9 in. cyclone on Fig 10, 7 of the runs ranged in underflow solid concentration from 24.1 to 49.0 pct with volume splits up to 8 pct to the underflow. There was no consistent displacement of these points and they lay both above and below the average performance line for the 9 in. cyclone.

To observe the effect under more severe conditions, three runs were made at a constant inlet diameter of 1.47 in. and overflow diameter of 1.625 in. Throughputs were held to a range of 112.1 to 118.7 gpm. During the first run, volume discharge to the underflow was maintained at a normal value to yield a high solid concentration. In the later runs, volume split to the underflow was severely increased resulting in a large dilution of the discharged solids. The results obtained are shown in Table 4.

Table 4 . . . Effect of Volume Split on Solid Elimination Efficiency

Run No.	Feed, gpm	Volume Pct to Underflow	Underflow Solid Concentration Weight, Pct	$\frac{(gpm)^{0.53}}{(eb)^{0.68}}$	50 Pct Point, Microns
1	112.1	5.6	59.6	6.75	11.5
2	118.7	12.5	36.6	6.94	10.5
3	113.3	15.8	35.5	6.77	10.25

The 50 pct points exhibited only a minor decrease with the increased volume splits indicating that no further concentrating action was taking place. Thus, the additional water directed to the underflow contained only the weight percentage of solids similar to that found in the overflow. Further proof of this is offered in Fig 14 where size analyses of the two streams are graphically portrayed. It will be observed that overflow size distribution experienced no appreciable change in consist while the underflow stream contained slightly greater quantities of solids below 11.5 microns for runs 2 and 3. If greater concentration was taking place, the overflow curves would have shown a general displacement towards the finer sizes for the later runs. A logical conclusion is that dilution of underflow solids beyond 45 to 50 pct offers no further assistance in the concentration of the feed solids. This appears theoretically feasible when one considers that no greater centrifugal force occurs with high volume splits to the underflow. Therefore increased concentrating action cannot be expected.

Type of Underflow Discharge

The vortex underflow is universally conceded to be superior to the overloaded discharge in regard to elimination efficiency.[5,6] When an overloaded discharge is in effect, the underflow nozzle is operating completely full of liquid and solids preventing the formation of an air core necessary for spiral

action. This results in a "pile-up" of solids at the apex of the cone and appreciable quantities are carried out in the overflow stream.

Perhaps the most striking demonstration of this inefficiency will be found in the data on the 14 in. Driessen cone at Kayford, W. Va., given in Table 1. Runs 6 and 7 were made under practically the same conditions except that an overloaded discharge was present in the latter case. It will be observed that 61.7 pct of the total feed solids were eliminated to the underflow in run 6 at a concentration of 58.3 pct. By contrast, run 7 eliminated only 7.3 pct of the total solids at a concentration of 65.5 pct. It is apparent that efficiency was very severely penalized while obtaining only a small increase in underflow solid concentration. Although pure overloaded discharge runs were not made with the 9 in. cyclone in this study due to the obvious inefficiency, previous tests had been performed with the 7 in. cyclone and were reported in an earlier paper.[6] Though the results were not as striking as the above case, all such tests indicated a serious injury to solid elimination.

The "sausage" discharge of Yancey and Geer could not be obtained in these studies due either to size consist or cyclone design.[5] However, something approximating this condition was obtained in certain runs wherein a very high underflow solid concentration up to 65 pct was experienced with a transition discharge. In these cases the 50 pct point was found to be approximately 25 pct above the pure vortex discharge with a corresponding injury to total solids eliminated. As almost the entire solids processed were below 200 mesh, the percentage of total solids rejected would have been higher if coarser material had been present. At the same time, from experiences encountered on the 14 in. Driessen cone it was felt that this compromise condition would have been very hard to maintain with the larger solids.

Weight Percentage of Solids in Feed

When the cyclone operates on slurries containing up to 15 pct solids, the factor of slurry weight will be of relatively minor importance. Specific densities of such feeds will be no greater than 1.10 in extreme cases. Furthermore, the critical density effecting the elimination efficiency will probably be closer to that of the overflow stream. Thus the difference between particle and slurry density, which is naturally of major concern in any liquid-solid separation, will decrease only a very small amount as solid concentration of the feed increases up to 15 pct. The cyclone will handle considerably more concentrated feeds but one must expect that slurry density will be of increasing importance in raising the 50 pct point.

In tests performed on the standard material, feed solid concentration ranged from 4.07 to 9.65 pct. In the elimination efficiency correlation of Fig 10, no consistent effect of solid concentration was apparent. As further indication of this, runs 6 and 7 on the 10 in. cyclone at Wilmington yielded almost identical results with regard to percentage of solids eliminated as exhibited in Table 2. These runs were made within one-half hour of each other after termination of coal movement through the tipple. The feed solid concentration of the first run was almost double that of the second and yet the difference between the percentage of solids eliminated was negligible. This was considered significant due to the existing similarity between size consist and quality for the two runs.

Particle Specific Gravity

As the particle specific gravity increases, still finer solids should be ejected to the underflow. Considering Stokes' law which applies to solids below the 200 mesh size,

$$w = \frac{(\rho_s - \rho)62.4gX^2}{18\mu} \qquad [14]$$

Therefore the diameter ratio of equal settling velocity particles possessing different gravities assuming the same slurry viscosity in both cases will be:

$$\frac{X_1}{X_2} = \left[\frac{\rho_{s2} - \rho_2}{\rho_{s1} - \rho_1}\right]^{1/2} \qquad [15]$$

If Stokes' law is valid for the prediction of cyclone 50 pct point for any particle gravity, then specific gravity parameter lines could be drawn parallel to the average performance line of Fig 10 and displaced from it according to Eq 15.

To validate this statement, a native ground barytes (barium sulphate, $BaSO_4$) of 4.30 sp gr was obtained for test purposes. Resulting elimination efficiency was plotted against the correlating factor on Fig 15 together with the average performance line of Fig 10. Also included is the predicted performance line for this material obtained from Eq 15 assuming a slurry density of 1.0. As pointed out in a previous discussion this will cause a negligible error.

It is apparent from Fig 15 that agreement between predicted and actual performance is excellent indicating Eq 15 is applicable to cyclone operation. Combining Eq 13 and 15 then,

elimination efficiency, microns

$$= 81\,\frac{(eb)^{0.68}}{(\text{gpm})^{0.53}}\left[\frac{2.73 - 1.0}{\rho_s - \rho}\right]^{0.5} \qquad [16]$$

Elimination efficiency data of Tables 1 and 2 were adjusted by use of Eq 15 to indicate the corresponding 50 pct point for a 2.73 gravity material. In these calculations, particle gravity was assumed to be the average of the overflow solids while slurry weight was taken as equal to the overflow density. These points were plotted against the correlating factor in Fig 16 and compared with the average performance line on actual 2.73 gravity material. Included are runs made on refuse slurries with the 9 in. cyclone.

In all cases but one, results were appreciably displaced upward from the average performance line. As the recycle water slurry used with the 10 in. cyclone contained large quantities of silt and clay, it was similar in nature to refuse material. For this type of slurry, an average performance line would appear to be about 25 to 30 pct higher than the standard line. Coal slurries of the 14 in. Driessen cone were higher but particular conditions surrounding the tests indicate that the four runs showing the greatest deviation should not be considered in this correlation. During these runs, solid concentration in the underflow ranged from 53.4 to 58.5 pct. Due to the very light average particle gravity of approximately 1.4, a more dilute underflow concentration of 50 pct solids or less is necessary to maintain a vortex discharge. With the underflow concentrations present in these runs, the transition discharge was instead in effect. Neglecting these runs, the deviation compares more favorably with the refuse slurries.

It is believed this deviation is due chiefly to the inability to accurately determine the 50 pct point. In the hydrometer analysis, an average solid specific gravity must be assumed and thus heavier materials present tend to increase the 50 pct size.

In spite of this error, the hydrometer method probably represents the best solution to the problem of size determination in the minus 200 mesh fraction of large gravity range particles. It offers a rapid and easy technique which is as accurate as any other procedure for this application. When predicting the percentage of total solids that could be eliminated by the cyclone for a particular slurry upon which a hydrometer analysis has been performed, a safety factor of 30 pct should accordingly be used in the determination of the 50 pct point by Eq 16.

Summary and Conclusions

Tests performed on the cyclone have proved it to be a simple and economic means for the elimination or recovery of fine solids from liquids in a sufficiently concentrated form so they may be further handled by ordinary equipment. Particles down to 9 and 10 microns have been eliminated from slurries. This dimension generally marks the bottom of the silt range and the top of the clay region indicating the cyclone's possibilities in not only the saving of fine coal but also the "cleaning-up" of water for reuse. In addition, operation is simple, initial cost is low, no moving parts are involved, and capacities up to approximately 350 gpm per sq ft of area can be obtained.

The following design features should be incorporated into a cyclone in order that solid elimination will be a maximum and energy requirements per feed gallon a minimum.

1. Maintain included angle of cone at minimum, preferably 15 or 20°.
2. Allow 6 to 10 in. between overflow point and conical section.
3. Underflow nozzle diameter should be large enough to permit a vigorous vortex discharge. This will be characterized by the presence of a vacuum at the underflow with a solid discharge concentration of 50 to 55 pct for refuse slurries and 50 pct or less for fine coal.

In calculating either the energy requirements or capacity of a cyclone, a formula of the type of Eq 8 is applicable. The gallons per minute used in the equation will be equal to the total feed rate minus the volume occupied by the ejectable solids in a 55 pct slurry. Eq 10 is applicable for a 20° included angle cyclone with less than 1 in. separation between overflow and conical section. If 6 in. or more separation is provided, the K value may be increased 6 to 8 pct and the expression will be valid for any cyclone with included angle of 20° or greater. Eq 11 may be used for any 15° cyclone.

The 50 pct point for any slurry may be predicted by use of Eq 16, using a safety factor of 30 pct in case a large particle gravity range is involved. The percentage of total solids that may be eliminated can finally be calculated from a screen analysis plus a hydrometer test on the minus 200 mesh fraction. All material coarser than the 50 pct point may be considered as ejectable.

In constructing the cyclone it is recommended that special heat treated alloys be used for certain parts in order to retard the rate of wear at critical points. All three nozzles, especially the underflow, are subject to abrasion and flanged parts should be provided at these locations in cyclone design. Nitriding of special alloys would probably offer the best solution to increased operating life.

Acknowledgments

The investigation was conducted in the Chemical Engineering laboratories of Northwestern University. Special acknowledgments are due Mr. R. L. Sutherland of the Truax-Traer Coal Co. and Messrs. James Jones and Melbourne McKee of the Northern Illinois Coal Corp. for their suggestions and cooperation in the study. The author also wishes to thank their respective companies for the opportunity of obtaining valuable test data. Finally, the following Northwestern University students have greatly aided the project by their help: George Andrae, Jerome Blau, John Graves, George Jandacek, Robert Piros, Arthur Robertson, Benjamin Schmetterer, Robert Slifer and Hubert Zaremba.

Nomenclature

b = inlet nozzle diameter, in.
e = overflow nozzle diameter, in.
F = cyclone energy loss, feet of fluid.
F' = gas cyclone energy loss expressed as feet of fluid flowing divided by inlet velocity head.
g = acceleration due to gravity, 32.2 ft per sec squared.
K = capacity ratio proportionality constant = $\frac{Q/\sqrt{F}}{(eb)^{0.9}}$
K' = gas cyclone proportionality constant = $F' \div (b/e)^2$
m = gas cyclone inlet diameter exponent = 2.0.
n = gas cyclone overflow diameter exponent = 2.0.
Q = gpm = gallons per minute of feed slurry (corrected as explained in case of capacity ratio determination).
s = liquid cyclone overflow diameter exponent, capacity ratio equation.
t = liquid cyclone inlet diameter exponent, capacity ratio equation.
v_i = inlet velocity, ft per sec.
w = settling velocity of particle, ft per sec.
X = diameter of particle, ft.
ρ = specific gravity of slurry, g per cc.
ρ_s = specific gravity of solid, g per cc.
μ = viscosity, lb per ft-sec.

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New York Talcs, Their Geological Features, Mining, Milling, and Uses

By A. E. J. ENGEL*

Introduction

The New York talc deposits of commercial importance are in St. Lawrence and Lewis counties, in the northwest Adirondack Mountains (Fig 1). All of the deposits are of pre-Cambrian age and occur within highly deformed and recrystallized marble of the Grenville series.

The deposits in St. Lawrence County, near Gouverneur, are the largest and most productive of their type known in the Western Hemisphere. In 1948 the seven mines which are in operation will produce about 130,000 tons of ground talc.

All talc production in Lewis County is from one mine. There the annual production ranges from 15,000 to 30,000 tons.

These so called talcs of New York State include earth materials of different chemical and mineral compositions. In general the mineral talc is subordinate in amount to other minerals in both the Gouverneur and Natural Bridge deposits.

In the Gouverneur district the mineral talc comprises less than 25 pct of the mined and ground rock. Most of the rock mined is a tremolite- or tremolite-anthophyllite schist somewhat altered to serpentine and talc.

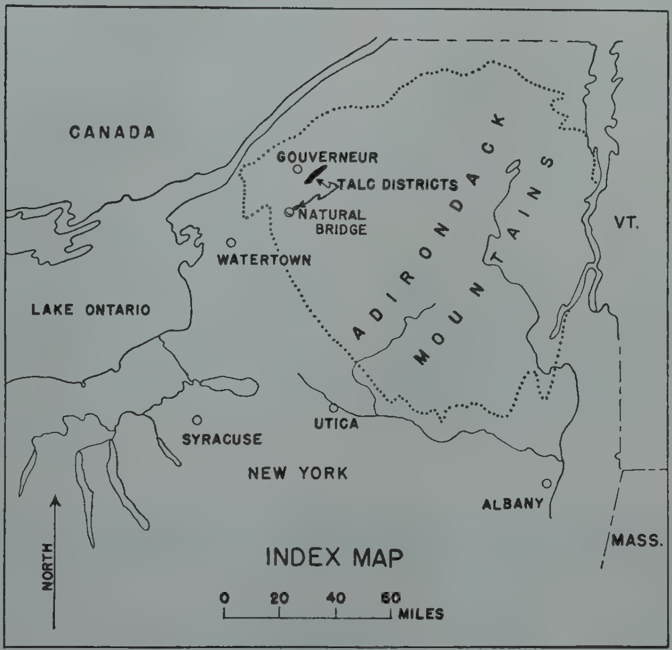

FIG 1—Map of New York talc deposits in the Adirondack Mountains.

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This characteristic in no way devaluates the "talc" for certain markets, and in some instances makes the material more desirable.

The Natural Bridge talcs include types high in serpentine, as well as complex aggregates of serpentine, talc, carbonates, and diopside.

Usage of the term talc, however, for

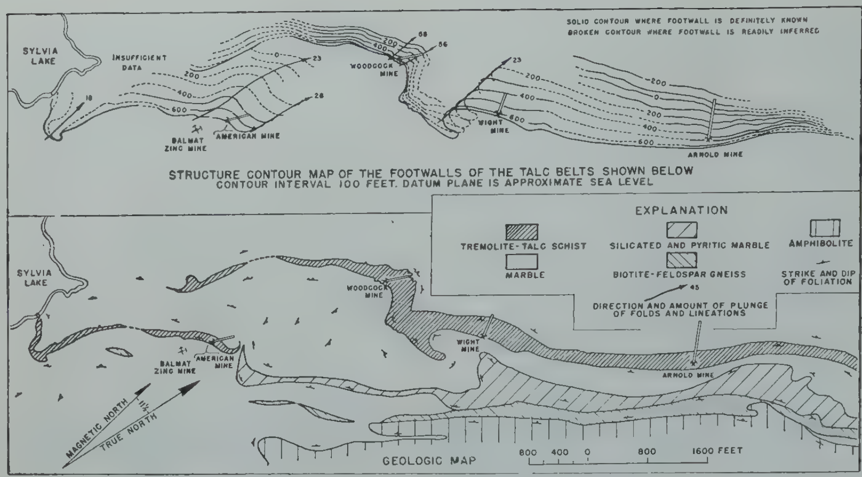

FIG 2—Geological features of the area north of Balmat, N.Y.

these industrial mineral aggregates is widespread and deeply rooted. Accordingly, this usage is followed in this paper. The phrase "the mineral talc" is used herein wherever reference is made to the specific mineral talc, $H_2Mg_3(SiO_3)_4$ or in terms of oxides $H_2O{\cdot}3MgO{\cdot}4SiO_2$.

General Geology

The talc of the Gouverneur district occurs in elongate zones interlayered within a northeastward-trending belt of impure marble.

The marble belt is apparently part of a highly deformed and metamorphosed flank of a northeastward-trending anticline. Cross folds, plunging north to northwest, foliations, shears, and lineations constitute important structural features of the talc zones (Fig 2 and 3).

The zones of commercial talc pinch, swell, and curve in sinuous to complexly folded patterns, Fig 2, but are rudely conformable with adjoining marble layers. The talc zones have a composite strike length of more than six miles, a probable extent down dip in excess of 2000 ft, and widths of as much as 400 ft. Dips along the talc belts are quite variable, ranging from the horizontal through the vertical, but averaging about 45° to the northwest.

Variations in thickness of the talc belts or of any included zone may be either abrupt or gradual. The belt near Talcville, which contains two producing mines, varies up to 300 ft or more in thickness, averaging perhaps 135 ft thick in the mines, Fig 3. Much of this thickness is commercial talc. A talc belt north of Balmat and southeast and east of Fowler, along which are 4 active mines, varies up to at least 425 ft in thickness, and averages possibly 125 ft thick, Fig 2. In this belt, however, one or several zones of commercial talc 6 to 25 ft thick or rarely as much as 75 ft thick are interlayered with impure or discolored noncommercial zones within the belt. Within these two belts are talc reserves sufficient to last several generations, at the present rate of production, under resourceful mining methods.

The approximate percentages of constituent minerals in various types of commercial talcs from the Gouverneur district are shown in Table 1.

A wide variation in proportions of tremolite, anthophyllite, serpentine, and talc are apparent. Other minerals which occur in and along the talc belts include quartz, calcite, dolomite, hexagonite (a manganese-bearing tremolite), iron and manganese oxides, diopside, chlorite, pyrite, mica, feldspars, titanite, magnesian- and manganese-bearing tourmalines, and apatite. Most of these last-named minerals constitute obvious adulterants or impurities and are avoided in mining.

The chemical compositions of some of the more important types of talc are indicated in Table 2. The range of variations in SiO_2 and MgO are readily apparent. Iron and manganese oxides, SO_3, and CO_2 when present in excess of the amounts shown constitute serious impurities for some markets. In general the companies in the Gouverneur district attempt to keep the CaO content between 3 and 7 pct, and the MgO content between 25 and 30 pct. The relatively high CaO content is largely a reflection of the lime present in tremolite and anthophyllite, Table 3.

Much of the tremolite probably formed by reactions between, and replacement of, favorable beds of quartzite and dolomite. This initial stratigraphic control of talc distribution was partly obscured, and to some extent superseded, by prominent secondary structures, especially shear zones that developed during metamorphism.

The talc-forming constituents doubtless were derived largely from the quartzite and dolomite beds, but water, silica, magnesia, and other elements were introduced into the present talc belts, and calcite removed by hydrothermal solutions.

Table 1 . . . The Approximate Percentages of Constituent Minerals in Layers of Commercial Talc and the Oil Absorption of These Layers

Specimen Number[a]	1	2	3	4	5	6	7	8	9	10
Tremolite	68	98	17		78	38	29	15	88	46
Anthophyllite			20				45	78	4	39
Talc, fibrous		1	63					5		
Talc, foliate					{>4}			1		5
Talc, shredded aggregate								1	1	
Serpentine, massive		1		80	18	54	21		4	{>4
Serpentine, fibrous							5			
Quartz	31	trace							2	4
Carbonates	1			2	trace	4				1
Hexagonite, iron and manganese oxides, mica, other impurities	trace			trace						1
Diopside				18		4			1	
Oil absorption 97.5 particles through 325 screen	29	33	58	56	45	50	55	52	35	48

[a] 1. Pale-pink, tremolite schist, hanging-wall side of talc belt, Talcville, N. Y.
2. Lustrous, white, stubby-bladed tremolite rock interlayered with Specimen No. 1.
3. Pale-gray to white, fibrous "talc," Talcville, N. Y.
4. Watery-green, serpentinized diopside rock along footwall talc belt, Talcville, N. Y.
5. Serpentinous tremolite, "10A ore," Talcville, N. Y.
6. Streaked, buff to chalky-tan, serpentinous tremolite, "regular ore," Talcville, N. Y.
7. Watery-gray, fibrous to bladed, tremolitic talc, Ontario mine, Fowler, N. Y.
8. "Arnold fiber," Arnold mine, Fowler, N. Y.
9. Medium-grained, serpentinous, splotched tremolite, "Arnold heavy stock," Arnold mine, Fowler, N. Y.
10. Pale buff, highly schistose "talc," hanging-wall zone, Woodcock mine, Balmat, N. Y.

Talc Mining

In general, methods used in mining New York talcs are less progressive than those used in the zinc mines of the same region.[1,2] Until recently, less than 30 pct of the commercial talc in the larger deposits was recovered. The amount of talc recovered from smaller, complex zones and ore bodies was considerably smaller. In general, however, and especially at Natural Bridge, N. Y., numerous natural obstacles in the way of efficient operations are presented by complexities of form, structure and composition of these talc deposits. At present, capable operators, who have introduced modern equipment, are in charge in most of the New York mines, and the projected plans of the several companies, if effected, will result in a more ideal exploitation of the important talc deposits.

Both the tabular deposits of the Gouverneur district and the brecciated talcose marble at Natural Bridge have a moderate to steep dip. In these deposits it is common practice to sink a shaft in the talc, on the footwall side of the desired rock. Since the commercial talc is followed by most of the shafts, changes in dip and plunge of the talc body are reflected in corresponding irregularities of the shafts.

One straight, inclined shaft was sunk in 1934 in the Gouverneur district, and another vertical, concrete-lined shaft is being sunk at present to an adjacent talc body.

In most of the New York mines, drifts and crosscuts driven from shafts are used to explore and outline the body of minable rock, as well as for tramways and subsequent mining needs. Since many deposits are of irregular, folded forms, many drifts are quite crooked. Almost no timbering is done in the Gouverneur district. At Natural Bridge, steel and concrete are employed in the shaft where it cuts a major fault zone, but the drifts are not timbered. Caving and bad ground constitute serious problems, however, in both districts.

In the Gouverneur district it is common practice in mining the talc bodies to drive raises at frequent intervals, often 30 to 50 ft, in whatever minable talc is encountered, as far as the overlying levels, or as far as safety or the upper limit of the talc body permits. Wet drilling is employed exclusively in the New York mines to reduce the hazard of silicosis and fibrosis.[3]

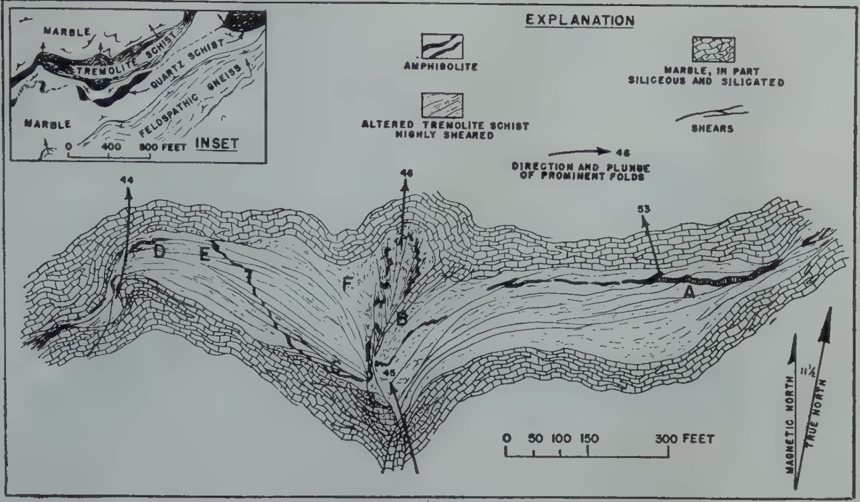

FIG 3—Simplified geological map of the seventh level of the International Talc Co. mine, Talcville, N.Y.

The raises are enlarged into stopes, with the broken rock falling to the drift floor to be mucked by hand or mechanically. A few old stopes in a number of New York mines "work themselves."

Recent practice in the Gouverneur district tends toward the development of conventional sublevel stopes wherever possible, with grizzlies and chutes. A valuable addition is a waterway with a fine-spaced grizzly placed in the intersection of appropriate raises and sublevels to drain off water which would otherwise soak the muck and impede milling (Fig 4). The underhand stoping (benching) method of mining commonly is employed where possible. In several mines in the Gouverneur district, blocks of ore are mined by long hole drilling.

Conventional practice in all the mines involves the use of air and electrically operated mucking machines, commonly rocker shovels, for mucking crosscuts and drifts. Small battery locomotives have supplanted hand tramming in all the mines. One company in the Gouverneur district sorts talc at the mine on steel plates in the head house. Under the steel plates is a series of bins into which talc is sorted according to color and impurities.

Milling

Talc milling in New York, as elsewhere, is largely a grinding operation, accompanied by air separation. Seven talc mills are in operation in New York, six of them in the Gouverneur district. Driers are employed in three of these mills in the Gouverneur district since

Table 2 . . . Chemical Analyses of Industrial Talcs Mined in New York State

Specimen Number[a]	1	2	3	4	5	6	7
SiO_2	59.80	66.23	67.0	56.50	59.40	57.26	47.0
Al_2O_3	0.57	1.05	}	1.0	0.74	1.14	1.71
Fe_2O_3	0.05	0.13	} 1.40	0.10	0.02	0.24	
FeO	0.15	0.22	}		0.12	0.05	
MnO	0.39	0.16	0.80		0.20	0.51	
CaO	6.80	2.26	2.30	6.20	4.94	6.50	6.61
MgO	27.45	25.71	24.80	30.40	30.09	29.08	33.5
TiO_2						0.04	
SO_3	0.07	0.01	0.07		0.01	0.14	
Ignition loss	4.75	3.86	3.10	4.80	4.09	3.98	7.74
Water (105°C)	0.45	0.25	0.30	0.77	0.47	0.34	1.96
CO_2	1.18	0.56	1.30	0.20	0.31	0.29	2.61
Total	101.66	100.44	101.07	99.97	100.39	99.57	101.13

[a] 1. Analysis of average sample of mined talc zone, Talcville, Gouverneur district, N. Y. Analyst Glen Edgington, U. S. Department of Agriculture.
2. Analysis of average sample of footwall talc zone, Fowler, Gouverneur district, N. Y. Analyst Glen Edgington, U. S. Department of Agriculture.
3. Analysis of hanging-wall talc zone, Woodcock mine, Balmat, N. Y. Analyst Orton Smalley, courtesy Loomis Talc Corp.
4. Analysis of footwall zone, American mine, Balmat, N. Y., courtesy St. Joseph Lead Co.
5. Analysis of "middle zone," Woodcock mine, Balmat, N. Y. Analyst Charles O'Brien, courtesy Loomis Talc Corp.
6. Analysis of average sample across commercial talc zone, Balmat, N. Y. Analyst F. A. Gonyer, Harvard University, courtesy R. T. Vanderbilt Co.
7. Analysis of "Micro Velva Talc," Natural Bridge, N. Y., courtesy Carbola Chemical Co.

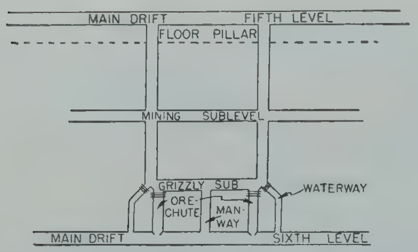

FIG 4—Diagrammatic sketch showing development of an inclined tabular or lenticular talc body by sublevel stopes.

wet material reduces mill capacity as much as 30 pct.

Talc taken from some of the mines may include blocks a foot or more in their largest dimensions. This rock goes directly from the mines to crushing plants at the mills, where primary and secondary crushers are used to reduce the talc to ½, 5⁄16, 3⁄16 or ⅛ in. products. Finer grinding is achieved in tube mills. Raymond mills, and Hardinge mills, in closed circuits with Sturtevant, Raymond or other type air separators.

Much New York talc is ground to one of the following general size groups: 97.5 pct particles through a 325 mesh screen, or 99.5 pct particles through a 325 mesh screen. Actually, in achieving either of these specifications, some talc is "overground" with a resulting large spread in particle dimensions. Many fines 15 to 35 microns in size commonly result.

Extremely fine grinding with fluid energy mills is done in several New York mills. A number of Micronizers and Wheeler mills are now in operation. Standard particle sizes ranging between one-half and 20 microns, as measured by air permeation methods, are attained in this fine grinding.

The following mill procedure is contemplated for a mill just erected in the Gouverneur district: primary crushing with gyratory crusher at shaft head house; conveyor to plant 500 ft away; wet storage; drying of minus ⅝ in. product; secondary crushing with Symons shorthead cone in closed circuit with ⅛ in. screen; dry ore storage; weightometer; extremely fine grinding with C. H. Wheeler Co. air mills and standard grinding with Hardinge mill in closed circuit with Raymond separator; Fuller-Kinyon conveying to bulk storage bins to packing house; Bates valve packers; power and gravity conveyors to cars.

Uses

Although New York talcs have a wide variety of uses, some 75,000 tons of talc from the state are employed annually by the paint industry. The fibrous form of the minerals anthophyllite and talc especially seems to hold heavier paint pigments in suspension longer, and to prevent caking and settling. The fibrous and blade-like mineral forms also are believed to serve as locking or bonding agents in the paint film. Better grades of Gouverneur talc have values of 90 or better in the standard industry whiteness scale.

A conventional standard for classifying New York talcs prepared for paints is the Gardner-Coleman oil absorption test.[4] In general, fine grinding increases the quantity of a given oil required to wet thoroughly all the absolute particle surface of the talc. For approximately uniform particle sizes, the talcs composed largely of the mineral talc, or serpentine, have a higher oil absorption than talcs rich in tremolite, quartz, or carbonates. Highly fibrous or flaky grains have greater oil absorption than equant grains of roughly the same size groups as indicated by conventional screening.

New York talcs range from 30 to 60 in the scale of oil absorption. The slightly serpentinous or talcose tremolite schist so common in the mines of the Gouverneur district tests between 40 and 45 when ground so 97.5 particles will pass through a 325 mesh screen. Essentially unaltered tremolite, ground to about the same size averages between 30 to 34 in oil absorption, Table 1. This same tremolite rock, ground to grain sizes of 1 to 25 microns, tests about 40 to 42 in oil absorption. Obviously the extent of surface area in relation to the mass of the grain is an important factor in determining oil absorption.

The ceramic industry used about 25,000 tons of New York talc in 1947, and probably will use a larger tonnage in 1948. In the manufacture of whiteware bodies, the CaO content of 4.5 to 6.5 which characterizes much Gouverneur talc is not at all objectionable, and in fact, may be desirable.[5]

Besides the consumption of New York talc in the paint and ceramic industries, appreciable tonnages are used in the insecticide, rubber, and textile industries.

In general, consumer demand has and continues to be for a talc of uniform chemical and physical properties. Accordingly, present philosophies in the two New York districts are to establish practical and efficient means of blending and averaging out the variations along and across the strike and down dip inherent in the talc deposits.

Table 3 . . . Approximate Composition by Oxides of the Common Minerals in, and Associated with, Industrial Talcs of New York

Mineral	CaO	MgO	Al_2O_3	SiO_2	CO_2	H_2O	MnO
Talc		32		63		3–7	
Serpentine		43		44		7–13	
Chlorite		36	18	33		5–14	
Tremolite	13	27		57		2	
Anthophyllite	3.5	31	0.5	59		2.5	3
Diopside	26	18		56			
Dolomite	30	22			48		
Calcite	56				44		
Quartz				100			

Acknowledgments

The writer's studies of New York talcs were undertaken for the U. S. Geological Survey, and welcomed and enlivened by the staffs of mining companies in the New York districts.

I also wish to acknowledge the inestimable contributions to my efforts made by H. M. Bannerman, C. N. Bozion, A. F. Buddington, James Page and many other members of the U. S. Geological Survey.

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What's New in Mining Safety

By J. J. FORBES* and S. H. ASH,* Member AIME

Introduction

Probably the newest thing in mining safety, or safety for mines, is the apparent dissatisfaction on the part of the mineral industries, as represented by both management and labor, and the general public of the Nation, with the safety record of the mining industry. This is reflected by published comments and papers in the technical press;[1–9] in labor periodicals,[10,11] in popular journals and periodicals,[12,13] and in the daily press; also by discussions of the Safety Committees sponsored by the National Safety Council,[9,14–15] the American Mining Congress,[1,16] the American Institute of Mining and Metallurgical Engineers,[2,4] the Mine Inspectors' Institute of America,[6,8,17,18] the Coal Mining Institute of America,[5,19] the American Standards Association, and others.[20,21]

Mining safety is a joint problem of management and the individual workman. Safety is not attained and will not be attained solely by laws, decrees, and commands because, if it could be done in this manner, acceptable mining safety would have been achieved long ago by the individual states in our Nation and by the nations abroad. The best safety performance is found at plants where management provides a safe environment for the workers; where supervision is efficient and supervisors, others concerned with promoting safety, and the individual worker know through training, experience, and personal contact the safe and unsafe practices of their field; and where the trained individual worker has become safety-minded by experience, quiet influence, unconscious suggestion, and personal guidance.

Despite what may be said to the contrary, progress has been and is being made in mining safety.[22–27] This is shown in Fig 1 to 4.

It is beyond the scope of this paper to give possible reasons for the valleys and peaks in the fatal-injury rates. Some recent factors that appear to be contributing to the improvement in these rates are briefly discussed.

Aside from mining equipment, "what's new in mining safety" for some mines has been in effect or was tried by others long ago. A safety program is a living thing; as such, it must be vitalized and continuously fed with enthusiasm, constructive action, and improvements. Because safety is concerned with human conduct, it is necessary to practice safety at all times; if not, it is soon forgotten, and accidents occur.

San Francisco Meeting, February 1949.

TP 2663 A. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Nov. 30, 1949. Manuscript received Jan. 28, 1949.

* Chief, Health and Safety Division, and Chief, Safety Branch, respectively, U. S. Bureau of Mines, Washington, D. C.

[1] References are at the end of the paper.

Mining-safety Records

The large employment and production concerned with anthracite contribute much to the mining-safety records of the mineral industry. Approximately 81,000 persons are employed in this industry having mines concentrated in an area of 480 square miles in northeastern Pennsylvania. Because of the achievement of the anthracite industry in safety during 1948, the other branches of the mineral industry can profit by a study of what has been done to achieve that record. Where one mineral is produced by other large groups having methods of mining that are similar in many respects to the anthracite-mining methods, an achievement in such a large industry is important.

Salient factors concerned with safety in the anthracite industry are: no radical change has occurred in the mining-safety regulations pertaining to anthracite mines. Because anthracite mining is a branch of the coal-mining industry,

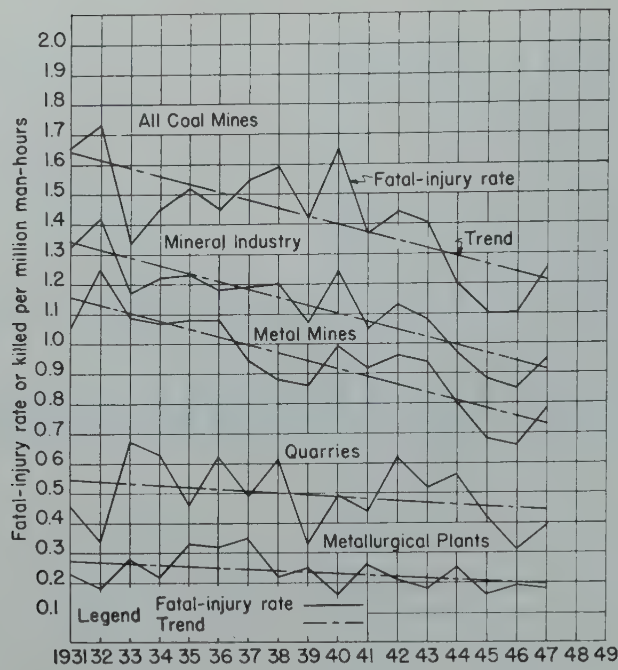

FIG 1—Fatal-injury rates and trends for mineral industry and its branches, 1931–1947.

it is important to know that the Federal code applying to bituminous-coal and lignite mines does not apply to anthracite mines; however, the anthracite industry and the United Mine Workers of America have made safety a part of the Joint Wage Agreement, and a health and welfare provision similar to that applying to bituminous-coal and lignite mines is included. These factors are new.

Moreover, it appears that, coupled with the foregoing factors, a monthly circular letter[27] from Joseph J. Walsh, Deputy Secretary of Mines, Pennsylvania, to all anthracite mine inspectors, mine officials, and miners has made a substantial contribution to the 1948 safety record of this industry.

Since mining began, the roof-fall hazard has been recognized as the No. 1 hazard in coal and noncoal mining. It still remains as such and is also the most difficult one to guard against.

By concentrating on this hazard, all concerned with safety in the anthracite industry have contributed the lowest annual fatality rate per million man-hours in the history of coal mining. This rate is 0.85. Of the 360 anthracite-mining companies, 320 had no fatalities for the 12-month period January to December 1948.[27] This record is new to coal-mining safety in the Nation. Because the improvement in the safety record is substantially the result of preventing injuries from falls of roof, and because available information on roof-fall injuries for the entire mining industry shows an upward trend for 1948, the anthracite record is significant.

NEW ACTION PROGRAMS AFFECTING MINING SAFETY

It is beyond the scope of this report even to mention safety programs and records of individual companies or mines;[4,28] likewise, it is not the purpose of this paper to point out poor safety records.

Because employment in bituminous-coal mining composes approximately 70 pct of the man-hours of exposure for the mineral industry, any safety program concerning the bituminous-coal-mining industry materially affects the fatal-injury rate and the severity rate of injuries for the mineral industry as a whole.[24]

It has long been recognized that voluntary employee cooperation and compliance with safety rules are indispensable requisites to improve the mining-safety record; moreover, experience has proved that where an industry acts as a unit to improve safety, the record improves.

The idea of making the promotion of safety in and about mines a responsibility to be shared equally by the employees and the management is not new. Neither is the idea new that this may be attained by law, because in March 1927 the legislature of the State of Washington passed laws that supplemented their code of mining laws and provided that safety committees (Sections 222–25, incl., 227, and 228, Chapter 306, Laws of Washington) be selected by management and labor to function at the coal mines of the State.[29] The foregoing code for mining safety by law resulted from action and agreement between the coal-mining industry and the United Mine Workers of America in the State of Washington

The idea of promoting safety in an industry by "wage agreement" is new and is typified by the inclusion of safety as part of the wage agreements both for the bituminous-coal and lignite mines of the United States and for the anthracite operators of Pennsylvania, and the United Mine Workers of America.

Because of the significant effect of employment on the injury rates for the mineral industry, salient points relating to safety in the bituminous-coal-mine agreement are found in a paper[5] given on Dec. 10, 1948, by C. F. Davis, Director of Safety, United Mine Workers of America, before the Coal Mining Institute of America:

> There is nothing new or revolutionary in the safety program of the United Mine Workers of America. Throughout the history of the miners' union, one of the chief objectives has been better mining laws, the enforcement of such laws when enacted, better and more competent supervision, and greater safety for men employed in the industry.
>
> For years we have realized that a comprehensive National Safety Program by our union was impossible unless Federal legislation was passed which would allow one set of rules, and a system of inspection that was uniform in all coal-producing areas. We have, therefore, consistently pressed for Federal legislation which would bring this about.
>
> The enactment of Public Law 49 by the 77th Congress,[30] providing for the setting up of standards and the inspection of coal mines by Federal mine inspectors, with written reports of such inspections to be sent to the company owning the mine, the mining department of the State in which the mine is located and the labor union (if any) having a contract at the mine, brought this program closer to realization. However, the fact that the law carried no enforcement provisions has hampered the mine workers in completely carrying out the objectives of the union.

The Wage Agreement of 1941 carried a provision for mine safety committee-

men, but their activities were restricted by the fact that recommendations of Federal mine inspectors were only advisory. State mining departments generally were noncooperative and coal companies in many instances were hostile to the idea.

This situation was greatly improved by an agreement entered into by Mr. John L. Lewis, President of the United Mine Workers of America, and Mr. Krug, Secretary of the Interior, representing the President of the United States, which provided for the setting up of a code of safety standards and their enforcement during the period of Government operation.

With the end of Government possession, the Federal mine safety code was made a part of the Joint Wage Agreement, and recognized nationally by both parties to the agreement as being the minimum safety requirements necessary to the industry, in the following language:

(A) *Mine Safety Code*

The Federal mine safety code[31] for bituminous coal and lignite mines of the United States, adopted pursuant to an agreement dated May 29, 1946, between the Secretary of the Interior and the President of the United Mine Workers of America and promulgated July 24, 1946, is hereby adopted and incorporated by reference in this contract as a code for health and safety in bituminous and lignite mines of the parties of the first part, with the following exceptions and alterations:

(1) The opening paragraph beginning with the words "pursuant to" and ending with the words "executive order" is stricken out.
(2) The words "coal mines administrator" are stricken out wherever they appear.
(3) Sections 5(a) and 5(b) of Article XII and all of Article XIV are stricken out.
(4) References in the code to its effective date shall be deemed to refer to the effective date of this contract.

(B) *Enforcement*

(1) Reports of Federal coal mine inspectors:—Wherever inspectors of the Federal Bureau of Mines, in making their inspections in accordance with authority as provided in Public Law 49, 77th Congress, find that there are violations of this code and make recommendations for the elimination of such noncompliance, the operators shall promptly comply with such recommendations, except as modified in paragraph two of this subdivision (B).
(2) Whenever either party to the contract feels that compliance with the recommendations of the Federal mine inspectors as provided above would cause irreparable damage or great injustice, they may appeal such recommendation to the joint board of review as hereinafter provided.

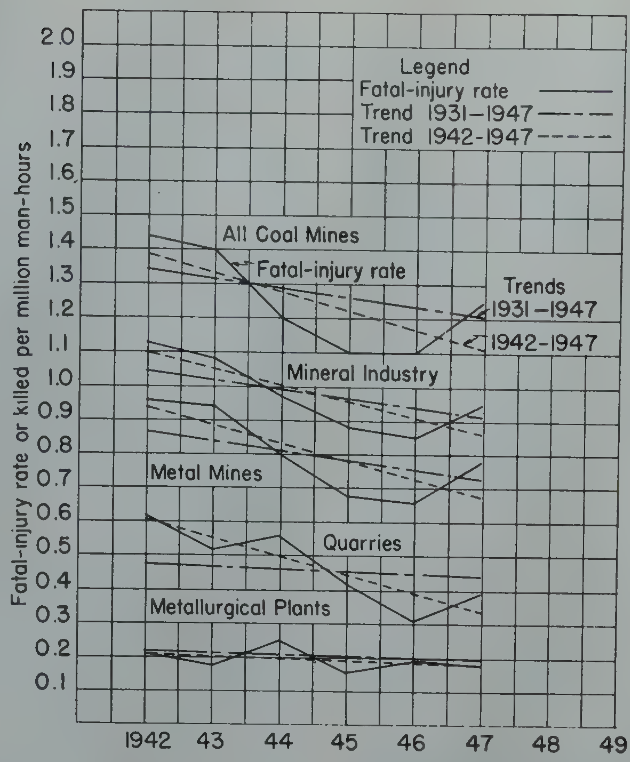

FIG 2—Fatal-injury rates and trends for mineral industry and its branches, 1942–1947, with 1931–1947 trends for comparison.

(C) *Review and Revision*

In order to carry out the intent and purposes of the agreement affecting the mine safety code, it is agreed that from time to time joint consultations shall be had with the U. S. Bureau of Mines looking toward review and appropriate revision of the mine safety code.

(D) *Joint Industry Safety Committee*

There is hereby established under this agreement a joint industry safety committee composed of four members, two of whom will be appointed by the mine workers and two of whom will be appointed by the operators, whose duty it shall be to (1) arbitrate any appeal which is filed with it by any operator or any mine worker who feels that any reported violation of the code and recommendations of compliance by a Federal coal mine inspector has not been justly reported or that the action required of him to correct the violation would subject him to irreparable damage or great injustice: and (2) to consult with the U. S. Bureau of Mines in accordance with the provisions of section (C) above.

(E) *Mine Safety Committee*

At each mine there shall be a mine safety committee selected by the local union. The committee members while engaged in the performance of their duties shall be paid by the union, but shall be deemed to be acting within the scope of their employment in the mine within the meaning of the Workmen's Compensation Law of the State where such duties are performed.

The mine safety committee may inspect any mine development or equipment used in producing coal. If the committee believes conditions found endanger the life and bodies of the mine workers, it shall report its finding and recommendations to the management. In those special instances where the committee believes an immediate danger exists and the committee recommends that the management remove all mine workers from the unsafe area, the operator is required to follow the recommendations of the committee.

If the safety committee in closing down an unsafe area acts arbitrarily and capriciously, members of such committee may be removed from the committee. Grievances that may arise as a result of a request for removal of a member of the safety committee under this section shall be handled in accordance with the provisions providing for settlement of disputes.

The safety committee and operators shall maintain such records concerning inspections, findings, recommendations, and actions relating to this provision of the agreement as may be required,

and copies of all reports made by the safety committee shall be filed with the operators.

In connection with section (D) of the contract let me say that any exception which has been allowed by the joint industry safety committee has been done after consultation with officials of the Federal Bureau of Mines and in all instances the board has followed the recommendations of the Bureau.

The safety department of the United Mine Workers of America studies reports of inspections made by Federal inspectors, calls violations of the mine safety code to the attention of the company, the district organization in which the mine is located, and the local union at the mine where the inspection was made.

We do not believe the code to be a perfect instrument. We know it could be improved. We believe, however, that compliance with all its provisions by all concerned, plus adequate supervision, would eliminate thousands of accidents each year.

We believe that accidents are caused; that they are the result of many contributing factors focused to a given point at a particular time; that the time to avoid accidents is before they happen, by eliminating the contributing factors.

We believe that the elimination of recognized hazards from the industry plus intelligent cooperation by all concerned could make the occupation of coal mining one of the safest in the Nation. Our entire efforts are directed at bringing this about.

The following provisions are abstracted from the Anthracite Wage Agreement between the International Union and Districts 1, 7, and 9, United Mine Workers of America, and the anthracite operators:

Mine Safety Program

(a) Federal Mine Safety Standards.

Inspectors of the Federal Bureau of Mines shall make periodic investigations of the mines, and report to the mine management and the United Mine Workers of America any violations of the Federal Safety Standards.[32]

Operators and Mine Workers agree to accept such standards of safety adaptable and practical to the anthracite industry, subject, however, to the right of review by the Director of the United States Bureau of Mines, upon petition from the Operator or the United Mine Workers of America. The right of review shall not delay taking of necessary steps to correct unsafe conditions where the immediate safety of men is involved.

Nothing herein shall operate to nullify existing State statutes, but this Agreement is intended to supplement the aforesaid statutes in the interest of mine safety.

(b) Mine Safety Committees.

Mine Committees, acting on request of individual employees or upon their own personal knowledge, shall have the right to report to management unsafe conditions affecting mine operations or equipment. In such cases the Mine Committee shall have the right, accompanied by representatives of management, to make necessary inspections of property or equipment for the ascertainment of actual facts. In the event such conditions, when reported, are not satisfactorily corrected, the Mine Committee shall request action by the State and Federal Inspectors.

Of the major branches of the mineral industry, quarrying has maintained the best safety record during the past 17 years. The quarrying industry, as represented by the Portland Cement Association, has proved that a safety program directed voluntarily on the industry level can accomplish much more than sporadic attempts by units of the industry throughout the United States.

An attempt, which is new, in a similar direction has been started by the National Coal Association's establishment of a Safety Division in February 1948, with the object of stimulating further interest in accident prevention among the member bituminous-coal-mining companies and the bituminous-coal industry as a whole.[19] The methods used by the association to achieve better safety are: (1) intimate contact by visitation with the heads of the producing companies and the men in charge of safety work; (2) acting as liaison between the member companies to disseminate information of methods and materials found to be successful in the individual companies for the use of all companies; and (3) sponsoring programs of accident prevention in the bituminous industry throughout the United States.

The National Coal Association believes that, through educational work as presented by the Federal Bureau of Mines in its first-aid and mine rescue training and accident-prevention classes for men and officials, much good can be accomplished.

Mining-safety Regulations

Mining-safety regulations in force in the United States can be divided into two general classes: mining-safety laws and mining-safety orders.

Changes in mining-safety laws are a slow process inherent in the procedure by which laws are made. Because mining-safety laws are changed with difficulty, it usually requires an aroused public opinion following a disaster to change them; this, coupled with the

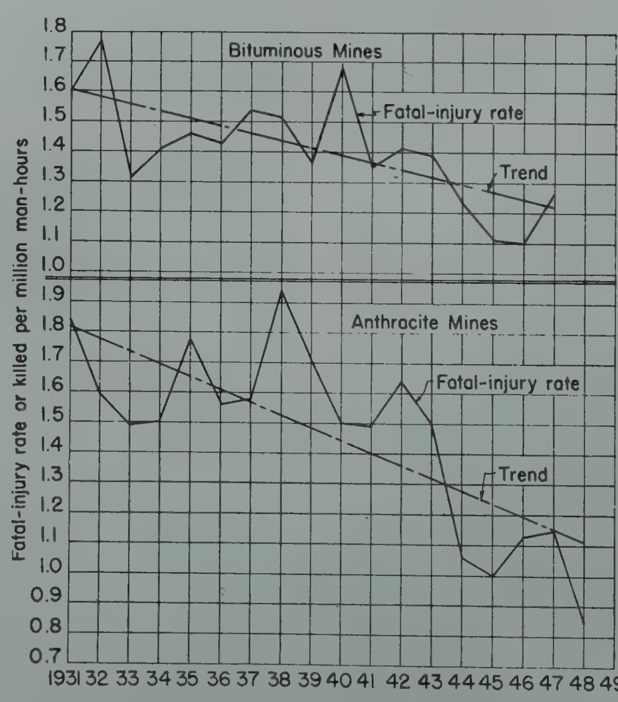

FIG 3—Fatal-injury rates and trends for bituminous-coal mines, 1931–1947, and for anthracite mines, 1931–1948.

resistance to change, as much as anything else is responsible for much of the dissatisfaction with many of our coal-mining-safety laws. On the other hand, mining-safety orders that have the effect of law can be changed in a satisfactory manner to meet changing conditions, new equipment, and progress. In general, the coal-mining states have laws; whereas, regulations pertaining to noncoal mines find more favor as orders.

Recent important changes in state coal-mining laws in Pennsylvania, Kentucky, Illinois, and Arkansas are discussed in detail elsewhere.[33] The changes in these laws are designed to prevent mine disasters.

A recent important change in mining-safety orders that is of interest to noncoal mining is for the State of New York which, in 1948, revised the Industrial Code relating to underground mining operations.[34] Those interested in mining-safety codes that are neither too cumbersome nor inequitable can well afford to study the process by which the Industrial Code in force for 28 years in this state is changed. The following "Foreword" appearing in the copy issued for "public critical review" is of interest:

The existing rules governing the mining and quarrying industries which are contained in Bulletin No. 17 of the Industrial Code have been in force for the past 28 years without any revision. During that time there have been considerable changes in methods and equipment used by these industries

Therefore, the Industrial Commissioner appointed an advisory committee consisting of representatives from the underground mining industry and other interested groups to review the existing rules and recommend such changes as they may deem necessary to bring this part of the Industrial Code up to date. After a series of conferences of this committee the following proposal was presented by them to the Industrial Commissioner as their recommendation. Their first proposal is to separate the two distinctly different subjects, underground mining and quarrying, and to place each under a separate cover; rules governing underground mining to be known as Rule No. 31. Quarrying and open-cut mining being sufficiently similar, it was deemed advisable to place these subjects under another separate cover.

The committee next proposed the elimination of obsolete rules such as those pertaining to animal haulage and open-flame permanent lighting, while proposed additions and revisions consisted of extensive changes in and additions to the entire code with particular emphasis on the rules governing electrical installations, use and handling of explosives, hoisting and haulage.

Finally, the rules as recommended by the advisory committee were grouped and numbered to conform with present-day practices.

The mining-safety laws of the Province of Ontario, Canada, were revised in 1948.[35] This code is one of the most comprehensive metal-mining codes that has been issued in North America and represents careful study of methods and equipment used in metal mines.

The most far-reaching endeavor to attain increased safety in the coal mines of the United States is embodied in the Federal system of coal-mine inspection. On May 7, 1941, a Federal system of coal-mine inspection was provided by an Act of Congress that empowers the Secretary of the Interior, acting through the Bureau of Mines, to "make or cause to be made annual or necessary inspections and investigations in coal mines the products of which regularly enter commerce or the operations of which substantially affect commerce."[30,36] The purpose of such inspections and investigations is to obtain information relating to health and safety conditions, to accidents causing bodily injury or loss of life, and to causes of occupational diseases originating in the coal mines of the United States.

Standards and rules pertaining to safety conditions and practices in bituminous-coal and lignite mines in the United States are embodied in the Federal Mine Safety Code.[31] This code is utilized by the Federal coal-mine inspectors for conducting inspections in bituminous-coal and lignite mines.

Federal Safety Standards,[32] applicable to anthracite mines, are utilized by Federal coal-mine inspectors when inspecting anthracite mines.

The Bureau of Mines administers the act through the Coal-Mine Inspection Branch, Health and Safety Division.[36] For the year 1948, funds were appropriated to employ a staff of 250 inspectors of various grades, 25 mining engineers, 5 mining-electrical engineers, 5 mining-explosives engineers, and 75 clerical assistants.

On appointment, new inspectors are trained at the Central Experiment Station of the Bureau of Mines, Pittsburgh, Pa. The training course comprises Code requirements, techni-

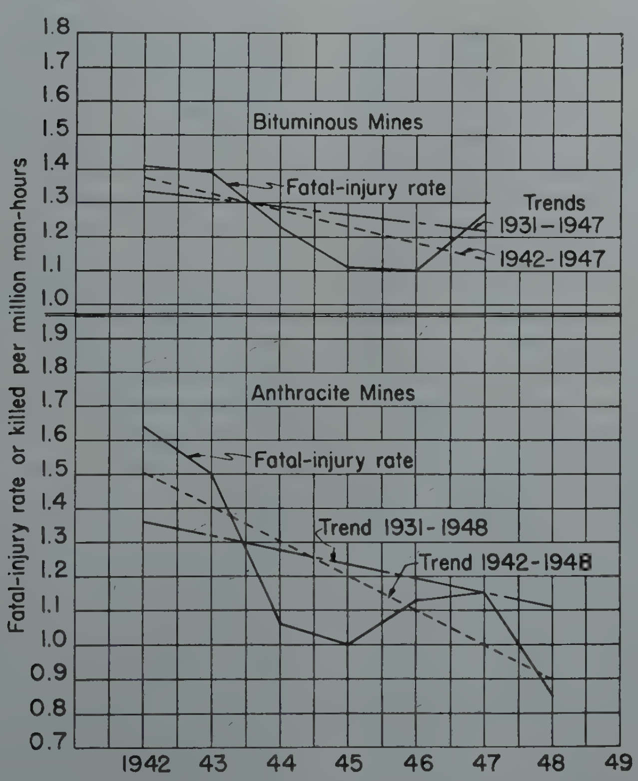

FIG 4—Fatal-injury rates and trends for bituminous-coal mines, 1942–1947, and for anthracite mines, 1942–1948, with 1931–1947 and 1931–1948 trends for comparison.

cal mine-safety subjects, mine rescue, first aid, inspection procedures, and safety organization.

Between 6000 and 7000 coal mines in the United States operate throughout the year, and approximately the same number produce coal in small quantities sometime during the year. Nearly 2500 of the continuously operating mines employ more than 25 persons each, and it was to these mines that the efforts of the Federal inspectors were directed first. Increases in the number of inspectors in 1947 and in 1948 have made possible the inspection of many of the small mines and more regular inspection of larger ones.

Cooperation is maintained between Federal and State inspectors.

In addition to their ordinary inspection duties, the Federal coal-mine inspectors address safety meetings of mine workers, write papers on safety subjects for publication, and undertake special investigations concerning particular hazards.

Research

Through observation and study of existent mining methods, of causes of accidents, of methods utilized in other industries doing similar tasks, and of the effects of injuries on the morale of workers and on the efficiency of the plant, researchers with ideas, open minds, and something to work with and strive for can find safer, easier, and cheaper ways of doing the job of modern mining.

Mass production of our mineral resources has required mass-production methods that, though increasing production, have created new hazards and, at the same time, removed others. Anything that will save labor, whether it is equipment or methods, can reduce injuries in mining. Borcherdt[37] has explained in detail the role of research in the development of new labor-saving equipment and methods.

Research that affects mining is both development and applied research. Pure research that may never reach the development and applied stages is necessary, and it is new to observe that segments of the mining industry are energetically putting the right foot forward in research.[1,7,10,17,25,26,37–42]

Remedies have been applied over the years to prevent injuries from falls of roof, which is the major cause of injuries in all branches of mining. More efficient supervision of the working faces,[18] systematic timbering,[43] and procedures for roof testing have been discussed time and again. It is obvious that the mining industry as a whole has not, as yet, found the proper remedy.[7,44,45]

Curzon[39] describes a research program comprising changing mining methods and blasting practices at the Holden mine[46] in recent years that has had a decided effect on preventing roof-fall injuries. Helms and others[47] describe research on the methods of large-scale stripping of steeply dipping beds in Pennsylvania's anthracite region. This, by reducing underground mining, has contributed materially to fewer roof-fall injuries in mining anthracite.

Thomas and others[17,45] have conducted research on the use of steel cross bars bolted to the roof or supported by pins in the ribs, as practiced in some metal mines. This has now been adopted in both coal and metal mines and is being tried in others.

In connection with size, spacing, and removal of pillars, research is now being conducted at Rifle, Colo., by the Bureau of Mines to determine the maximum safe width of rooms. Obert and Duvall[48] are conducting tests in mines in the iron ranges to detect overloads on pillars by use of recording geophones. This contributes knowledge on rock bursts.

Research is being conducted by the Bureau of Mines, the General Reinsurance Corp., and the Mine Safety Appliances Co., on a new electronic roof-testing device. Indications are encouraging, in that this method of roof testing will be applicable to coal roof as well as to all other types of mine roof and that this new method of testing eventually will give the mining industry a practicable and positive means of determining roof conditions and will thus reduce the hazard to which most mining fatalities can now be attributed. Continuance of this research should very definitely contribute to the solution of the problem of obtaining adequate mine-roof detection.

A large percentage of explosives manufactured in the United States are utilized by the mineral industry. Hazards to life and property are present when explosives are used for mining. Bickel[15] has recently described blasting hazards in metal mines, and research conducted to obviate such hazards.

During 1946 to 1947, research was conducted in some large coal mines in the State of Washington by representatives of the State Mine-Inspection Department, the mining companies, the explosives companies, and the Bureau of Mines on the use of fast-delay electric blasting caps. It was found that safer and more efficient blasting is obtained when compared with any other system of single-shot or multiple blasting.

Observations concerning fast-delay electric blasting caps are: fast-delay or millisecond electric blasting caps, introduced over 3 years ago, fire individually but in such small time relation to each other that the action of certain charges supplements the action of other charges before there has been any appreciable movement of the burden or release of gases from behind the burden. Whereas regular-delay electric blasting caps have intervals between firing periods that are measured in seconds and single-shot blasting necessitates intervals between firing periods measured in minutes, the fast delays are measured in milli- or thousandths of seconds. Although the firing of individual regular delays in a series is perceptible to the ear, that of the millisecond delays is too rapid for reliable perception. Such small intervals may best be recorded with some form of electrical timing device.[49]

When fast-delay electric blasting caps are used in burden that is not broken up by cracks and fissures that would permit the gases of the explosion to escape before they have done their useful work, better fragmentation, reduced back break, less vibration, less smoke, reduced explosive gas, less dust, and controlled throw are attained.

Research on explosives by the Bureau of Mines is continuous. More than 1600 tests have been made on permissible explosives, blasting devices, special types of explosives, detonators, and hazardous chemicals. Tests included the effects of sheaths on gaseous products from permissible explosives, the effect of oxygen balance of explosives on gaseous products, the explosion hazards of perchloric acid and mixtures with organic materials, and ignition hazards of mixtures containing ammonium nitrate.

The Bureau of Mines has approved explosives and blasting units classed as permissible since 1909 (explosives) and 1924 (blasting units).[50,51] Because of improvements in explosives, on July 20, 1946, the Bureau approved both the extension of the 1½ lb limit for using permissible explosives to a 3 lb limit and the use of permissible multi-

ple-shot blasting units.[52]

Research has been conducted by the Bureau of Mines in the laboratory and in the field on diesel-powered locomotives and engines.[53,54] Information has been obtained on the quantity of harmful and objectionable gases produced by diesel engines and on the hazards of operating them in normal atmospheres and in explosive atmospheres. Methods to prevent ignition of explosive atmospheres were determined. A schedule of requirements that must be met by diesel locomotives that are permissible for safe use in coal mines was formulated in 1944, and in 1948 a schedule was prepared on their use in mines where explosive gases are not encountered. Future work contemplates research on methods to condition the exhaust gas so as to minimize the discharge of harmful exhaust gases.

In cooperation with manufacturers and mine operators the Bureau of Mines has participated in the design and development of new coal-mining equipment to meet difficult and unusual mining conditions, to increase safety, and to conserve coal. Shearing machines and coal planers in connection with loading units have been tested successfully. Utilization of such machines can reduce numerous hazards at the faces.[55,56]

A new continuous mining machine, which combines the operations of cutting, drilling, shooting, and loading, was introduced to industry by the Sunnyhill Coal Co., at its plant in New Lexington, Ohio, on Oct. 27, 1948. Although the company feels that the machine has demonstrated its practicability, they still regard it as somewhat in the development stage as far as actual operating methods are concerned.[38] Another continuous mining machine, developed by the Joy Manufacturing Co., was introduced to industry on Dec. 15, 1948, in the Mathies mine of the Pittsburgh Coal Co., Finleyville, Pa.[10]

Boreholes of different diameters for connecting advance workings to the surface for ventilation, communication, prospecting, power circuits, and escapeways have solved problems in safety and development. Originally used in metal and other noncoal mines, they are being found useful in coal mining where the depth is not too great. The boreholes range from 6 to 48 in. in diameter, the larger ones often fitted with a hoist and man cage.[57–59]

A recent example at a coal mine is an escape shaft 48 in. in diameter and 200 ft deep. The shaft serves as an outlet for return air, 50,000 cu ft a minute being exhausted. A steel cage for six men is operated by a gasoline-powered hoist. Telephone communication is installed.

Investigations of the utilization of radio for communication underground have been continued at intervals since 1920. Recent experiments indicate voice communication by radio through soil and strata is feasible and may be applied in some normal mine operations. The problem requires further investigation and adaptation of equipment, but the results so far are encouraging.[60,61]

Haulage accidents comprise the second large group of mine accidents. Haulage jobs are the most dangerous class in mining. Despite these facts, haulage looms very large as needing effective action in mining safety.

Despite numerous safety devices and research conducted to find a positive dependable safety device to prevent disasters by falling cages and cars, disasters happen from time to time. Because of the Paymaster mine accident on Feb. 22, 1945, when 16 men were killed when the hoisting cable failed, the most comprehensive research program regarding hoisting equipment and hoisting practice ever conducted in America was made for Ontario mines by a committee appointed by the Province of Ontario. The report of this committee in 1947[42] is an invaluable contribution on the foregoing subject.

Mining-safety research is a problem throughout the world and, after a lapse of 7 years, the Fifth International Conference of Directors of Mine-Safety Research was held at Pittsburgh, Pa., on Sept. 20–25, 1948. Representatives from Great Britain, Belgium, France, Germany, Poland, and the United States participated in the program, covering subjects on mining safety.[62]

Research on means of reducing the hazards of coal-mine explosions is continuous by the Bureau of Mines. Many existent methods are improved and new ones developed. The utility of different materials and methods for extinguishing mine fires is investigated. Studies are continuous on the mechanism of ignition, propagation of flame, and flame quenching to further protect against ignition of explosive gas in mines.[50,63–65]

Safety Education

There are as many different opinions regarding what constitutes safety education as there are as to what constitutes good supervision. An effective program of safety education must not only focus the attention of the workmen and officials on accident prevention, but it must also secure their interest and desire to do everything possible to prevent accidents, or the program is doomed to failure.

Auburn[14] has ably explained that the employee is the focal point in the prevention of accidents and that an accident-prevention program must be planned around him.

The Coal Division of the American Mining Congress has set up a program on safety education.[1]

The part that job training must play and can accomplish in safety education is more recently proved by job-training programs that get results in widely separated localities.[14,16,40,66–69]

Accident-prevention courses of the Federal Bureau of Mines are available for supervisors and workmen alike to become familiar with the store of mine-safety technology that has been developed and accumulated by this Bureau through its close contact with safety problems of the mineral industry during the 38 years of the Bureau's existence.

The text of the course for bituminous-coal mines covers the subjects of accident statistics; falls of roof and coal; hoisting and haulage; explosions and fires; explosives; electricity; and miscellaneous hazards. The course is published as Miners' Circulars 47 to 50, and 58 to 60.

The text of the course for metal mines covers the subjects of accident statistics, falls of rock or ore; hoisting and haulage; explosives; fires, gases, and ventilation; electrical and mechanical hazards; and health and miscellaneous hazards. The course is published as Miners' Circulars 51 to 57.

A similar course is prepared and given for the petroleum industry. Following research begun in 1932 and ended in 1947, a bulletin[70] on safety practices in dredging and hydraulic mining has been published and constitutes a text for this branch of mining.

Research is being conducted by the Bureau at smelters, refineries, and processing plants so as to obtain information to serve as an accident-prevention course for the metallurgical industry.

Copies of the above-mentioned publications are given free to all persons who enroll in the respective courses.

The Bureau's courses are based upon general conditions found in the mineral industry, and broad phases of safety are discussed in general terms. Safety and production are closely related, and the courses are designed to bring the two into proper focus.

The successful instructors of these courses are experienced engineers who devote most of their time and discussions to the particular conditions found in the mine or plant where the class is being conducted.

In addition to the miners' circulars previously mentioned, the Bureau of Mines has made available other facilities to assist the instructors in presenting the course. Numerous lantern slides have been prepared from mining photographs, diagrams, and maps illustrating various topics in the course, and each instructor has a complete set with a slide projector for displaying them.

This store of visual educational aids is augmented from time to time by the addition of slides depicting up-to-date developments in mining. Each instructor is provided with a gas-explosion gallery in which he can demonstrate the ignition of methane by arcs and sparks, as well as the effect of black damp or low oxygen content in methane-air or other gas-air mixtures. A coal-dust explosion gallery has been provided for each instructor to demonstrate both the explosibility of bituminous-coal dust and the effect of rock dust in preventing the ignition of coal dust and propagation of flame. Each instructor has equipment useful for class-demonstration purposes.

To date only a small percentage of the total number of persons employed in the mineral industry has been given these courses; but the number of courses conducted each year is increasing, and requests for repetitions of the courses are indicative of interest in them. A certificate is awarded by the Bureau of Mines to all persons who complete the course of training.

After 40 years of continuous effort to decrease accidents in the mineral industry, the Bureau of Mines continues to believe that first-aid training is the best foundation upon which a safety program can be built. For nearly 15 years before World War II, approximately 10 pct of the total number of persons employed in the mineral industry were trained annually through a cooperative program of mining companies and the Bureau of Mines. During the war, first-aid training was neglected for what appeared to be good reasons at the time, and for two years (1945 to 1946) only 2 pct of the total number of persons employed in the mineral industries were given first-aid training.

The demands from the mineral industry for more intensive first-aid training of personnel has led to development of a new phase of cooperative training. Most operating companies have in their employ qualified first-aid instructors whose ability to teach has been well-demonstrated. These instructors now bear the brunt of the work and are responsible to their employers and the Bureau of Mines for maintaining a standard of teaching worthy of a Bureau of Mines certificate.

Under the foregoing program, the limited personnel of the Bureau is able to supervise and assist in the training of a much greater number of men than could otherwise be possible. The number of persons trained in first aid has doubled in the last two years; and it is expected that, with the growth of this type of training, the usefulness of Bureau personnel can be extended as never before. Men holding Bureau of Mines first-aid certificates develop a sense of belonging to an organization that is seriously dedicated to the prevention of accidents. If a sense of safety-consciousness can be embedded and maintained in each worker, the number of accidents will be decreased.

Conclusion

The battleground for the world today is at the worker level.[71] The battleground for mining safety has always been at the worker level, and so we need among our leaders for mining safety men and women who know our mining people best and are closest to them. Safety trends to the degree that this fundamental fact is recognized in action programs.

Just as long as mine workers are expected to continue to use individual judgment on methods of doing work, and those charged with responsibility for mining safety endeavor to fix responsibility for accidents either on the implied carelessness of the worker rather than enlist his services in action programs or on the implied callousness of the employer, improvement in mining safety will lag. Furthermore, because a worker today may become an immediate supervisor tomorrow, the safety-consciousness inherent in this worker will be that on which he, as a supervisor, must grow.

Improvement in mining safety is evident in those plants and branches of the mineral industry where the workers are an integral part of the safety program.

Labor leaders can bring special equipment to an assignment like mining safety. They are used to dealing with people.[71] Unions must support all constructive efforts to prevent accidents. They must, where it is indicated, side with management in the correction of faults of the worker for his own safety.[14]

The improvement in mining safety lies with the industry. Something must be done to get the attention of the worker in mining safety, create his interest in it, and maintain a firm desire on his part to practice safety. The worker and supervisor alike must realize and believe that the worker is the focal point in the prevention of accidents.

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A New Method of Weighting Core and Cuttings in Diamond Drilling

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To evaluate chemically the sample of rock obtained by diamond drilling, it has long been recognized that the analyses of the two components of the sample, core and sludge, must be given appropriate influence in computing the average analysis of any unit of depth. The purpose of this investigation is to set forth what means are available for apportioning the effect of core and sludge on the final analysis, what variables affect the problem, and what combination of applied mathematics will closest approximate the truth under each condition as these variables proceed within their limitations.

A drill hole is bored in iron ore exploration principally to test variations in rock composition with depth and is usually directed as nearly normal to the bedding of a horizon to be tested as possible. This practice has a tendency to minimize variation in composition laterally which in any event is not likely to be great. It is obvious that the opportunity for change in analysis of a particular rock is not statistically as great radially in a diamond drill hole where the distance in which such a change may occur is from 0.719 in. (EX bit) to 1.469 in. (NX bit) as there would be longitudinally even in a run as short as 5 ft. Variations in composition of bedded or layered rocks are usually greater normal to the bedding than parallel thereto. Even in massive rocks, like porphyries, variations are functions of distance. Hence, in either case, variations along the hole are of greater effect than across it.

Let us examine Fig 1 briefly to observe a cross section of a unit of depth of a typical diamond drill hole.

If we neglect radial change in chemical composition, which we have seen is small compared to lengthwise variation, we can see that if core recovery was 100 pct, the core and its surrounding area, which would be recovered as cuttings (assuming 100 pct sludge recovery), must analyze the same if the sampling is perfect. This is the foundation of this paper, namely, that if core recovery is complete we may assume that the core constitutes as nearly as possible a perfect sample of the ground drilled.

Now let us pass to the methods of weighting core and sludge in the final analysis and then develop each in turn. One might employ core analyses only, sludge analyses only, or apportion the influence of each by one of several methods. The most common apportionment is by direct proportion to relative theoretical volume occupied by each component in the cylinder hollowed out by the drill, known as the Longyear formula.

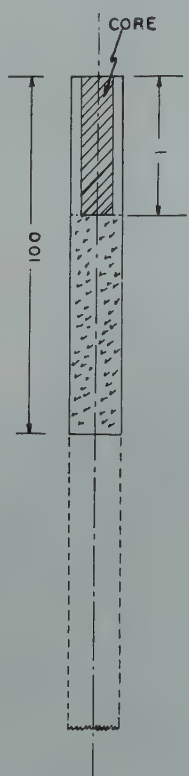

FIG 1—Cross section of diamond drill hole.

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Another logical treatment would be to consider core and sludge in the final analysis according to the weight of each recovered. This procedure was elaborately set forth and refined by R. S. Moehlman.[1] Then to be sure that we have embraced all the possibilities, we must admit that some other method or methods may be devised empirically or by mathematical maneuvers.

Should a drill machine be capable of recovering 100 pct of the core in all types of rock, certainly core alone need be analyzed and no complicated mathematics are required; but although some companies have had fair success in obtaining high percentages of core in certain homogeneous ores on the Marquette Range, their same methods have proved disappointing on the Menominee, Cuyuna, and Gogebic Ranges where the iron formation often varies widely from flinty chert beds interstratified with soft hematite to wholly leached ore material composed of an unpredictable mixture of hematite and limonite with occasional zones of sugary, recrystallized chert. In spite of numerous mechanical improvements in core barrels, few drilling programs can count on complete core recovery.

The sludge alone might be analyzed but for practical reasons this is inadvisable when core is obtained due to the numerous opportunities for contaminating or losing portions of the cuttings.

The Longyear formula involves a weighting of the core and sludge in proportion to the theoretical volume of each, that is, where the volume of the core recovered is one-third of the total cubic volume of the cylindrical hole made by the drill, the analysis of the core is given one-third of the weight in the final analysis. This approach to

[1] R. S. Moehlman: Diamond Drilling in Exploration and Development. *Trans.* AIME (1945) 163, 491–510.

the question has certain disadvantages, particularly in the high core recovery brackets. Even when 100 pct of the core is recovered, the volume of the core is only from 35.5 to 51.2 pct of the volume of the hole, depending on the bit size, and is weighted accordingly. This runs counter to our basic principle that when all the core is recovered it should receive all the weight in the final analysis.

Another often used apportionment is by proportion to the relative weight of core and sludge recovered. This method has proved particularly valuable in the Iron River District when considerable sludge has been lost, and the remainder may be less reliable as a sample of the material ground up. However, the same weakness is found as in the Longyear formula in the regions of high core recovery.

From the above discussion, it seems apparent that some formula must be developed which, while it conforms with volumetric proportions in the lower reaches of core recovery, gradually transfers emphasis to core as recovery mounts. Therefore we are constrained to look into the mathematics of core weighting.

First let us set up the algebraic terms involved:

Let At = the final average analysis of the whole run.

Let A = the analysis of the sludge in the zone which did not core. (This zone is checked in Fig 1.)

Let C = the analysis of the core, as recovered from the zone marked i in Fig 1.

Let S = the analysis of the total sludge both blank and checked areas included. (This is the sludge analysis normally provided by the chemist and is from all or a portion of the whole 100 linear units considered in this analysis as shown in Fig 1.)

Now we are ready to analyze the volumetric or Longyear method of weighting:

Let a = the figure for percentage of the total volume of the hole occupied by core when core recovery is 100 pct. This figure varies between 35.5 and 51.2 and is a function of the relation between the second power of the outside and inside diameters of the particular bit in question.

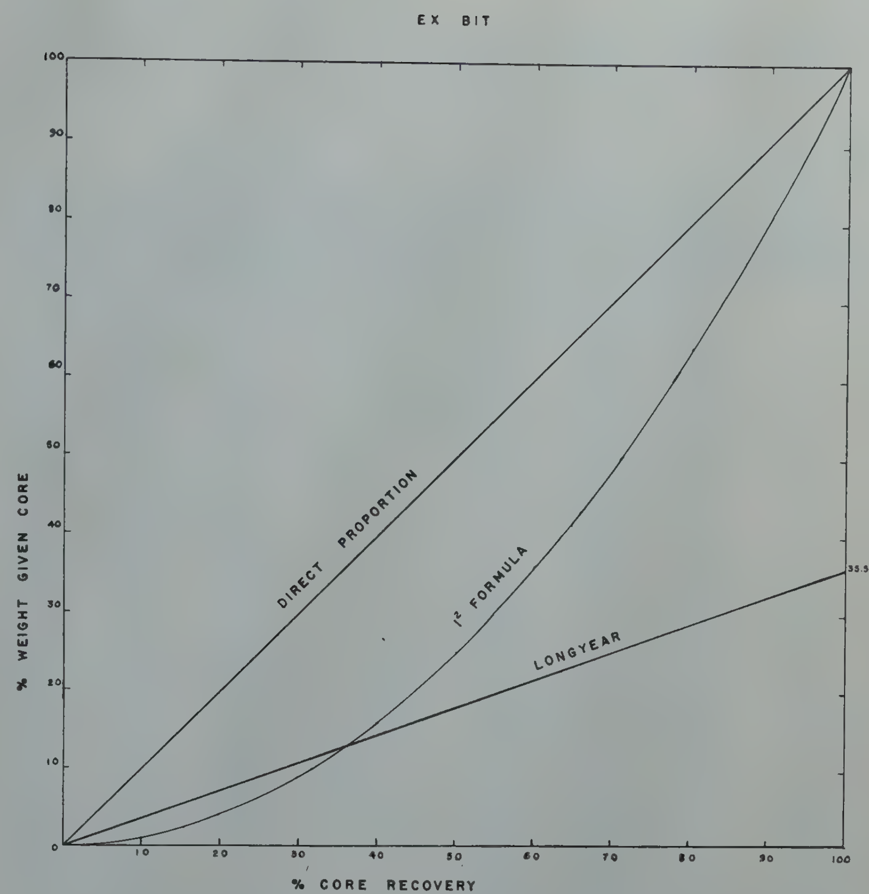

FIG 2—Comparison curves of the direct proportion, i^2, and Longyear formulas.

Let i = percentage of core recovery.

1. Percentage of total volume in checked area in Fig 1 = $100 - i$

2. Percentage of total volume represented by core = $\frac{ia}{100}$

3. Percentage of total volume in blank area in Fig 1 (sludge which analyzes the same as the core) = $i - \frac{ia}{100}$

4. The percentage of the total volume in the entire sludge = $100 - \frac{ia}{100}$

5. Therefore,

$$S = \frac{\left(i - \frac{ia}{100}\right) C + (100 - i)A}{100 - \frac{ia}{100}}$$

6. Solving for A we find,

$$A = \frac{S\left(100 - \frac{ia}{100}\right) - \left(i - \frac{ia}{100}\right) C}{100 - i}$$

The value of A is quite academic but it does represent the theoretical analysis of the area which did not core.

7. $At = \frac{iC + (100 - i)A}{100}$

8. By substituting the values for A we obtain:

$$At = \frac{S\left(100 - \frac{ia}{100}\right) + \frac{ia}{100} C}{100}$$

This last is the algebraic basis of the Longyear factors, the figures in the table being equal to $\left(100 - \frac{ia}{100}\right)$ or $\frac{ia}{100}$ depending on whether the sludge or the core factor is sought.

Core and sludge influence apportioned according to the weight of each recovered may be expressed mathematically as:

$$At = \frac{SWs + CWc}{Ws + Wc}$$

where Wc and Ws are the weight of core and sludge recovered, respectively. It might be well to note here that this formula cannot be represented graphically since it involves unpredictable variations in core and sludge recovery.

It is suggested that the core and sludge might be weighted directly according to the percentage of each obtained, which would be expressed:

$$At = \frac{S(100 - i) + iC}{100}$$

(See Fig 2 for graphic representation.)

It can be seen that in this direct proportion if core recovery is slight, the core still has a strong influence in the final analysis so that when anomolous beds only are cored, it might prove to give undue weight to the core.

Table 1 . . . Multipliers Derived from the i^2 Formula, to Be Used in Combining Analyses of Core and Cuttings to Obtain Final Analysis of the Drill Run

Inches of Core	EX		AX		BX		NX	
	Core Factor	Sludge Factor	Core Factor	Sludge Factor	Core Factor	Sludge Factor	Core Factor	Sludge Factor
1	0.6	99.4	0.6	99.4	0.8	99.2	0.9	99.1
2	1.2	98.8	1.2	98.8	1.6	98.4	1.7	98.3
3	1.8	98.2	1.8	98.2	2.4	97.6	2.6	97.4
4	2.4	97.6	2.4	97.6	3.2	96.8	3.4	96.6
5	3.0	97.0	3.0	97.0	4.0	96.0	4.3	95.7
6	3.5	96.5	3.6	96.4	4.8	95.2	5.1	94.9
7	4.1	95.9	4.2	95.8	5.6	94.4	6.0	94.0
8	4.7	95.3	4.8	95.2	6.4	93.6	6.8	93.2
9	5.3	94.7	5.4	94.6	7.2	92.8	7.7	92.3
10	5.9	94.1	6.0	94.0	8.0	92.0	8.5	91.5
11	6.5	93.5	6.6	93.4	8.8	91.2	9.4	90.6
12	7.1	92.9	7.2	92.8	9.6	90.4	10.2	89.8
13	7.7	92.3	7.8	92.2	10.4	89.6	11.1	88.9
14	8.3	91.7	8.4	91.6	11.2	88.8	11.9	88.1
15	8.9	91.1	9.0	91.0	12.0	88.0	12.8	87.2
16	9.5	90.5	9.6	90.4	12.8	87.2	13.7	86.3
17	10.1	89.9	10.2	89.8	13.6	86.4	14.5	85.5
18	10.6	89.4	10.8	89.2	14.4	85.6	15.4	84.6
19	11.2	88.8	11.4	88.6	15.2	84.8	16.2	83.8
20	11.8	88.2	12.0	88.0	16.0	84.0	17.1	82.9
21	12.4	87.6	12.6	87.4	16.8	83.2	17.9	82.1
22	13.5	86.5	13.5	86.5	17.6	82.4	18.8	81.2
23	14.7	85.3	14.7	85.3	18.4	81.6	19.6	80.4
24	16.0	84.0	16.0	84.0	19.2	80.8	20.5	79.5
25	17.3	82.7	17.3	82.7	20.0	80.0	21.3	78.7
26	18.8	81.2	18.8	81.2	20.8	79.2	22.2	77.8
27	19.4	80.6	19.4	80.6	21.6	78.4	23.1	76.9
28	21.8	78.2	21.8	78.2	22.4	77.6	23.9	76.1
29	23.4	76.6	23.4	76.6	23.4	76.6	24.8	75.2
30	25.0	75.0	25.0	75.0	25.0	75.0	25.6	74.4
31	26.7	73.3	26.7	73.3	26.7	73.3	26.7	73.3
32	28.4	71.6	28.4	71.6	28.4	71.6	28.4	71.6
33	30.3	69.7	30.3	69.7	30.3	69.7	30.3	69.7
34	32.1	67.9	32.1	67.9	32.1	67.9	32.1	67.9
35	34.0	66.0	34.0	66.0	34.0	66.0	34.0	66.0
36	36.0	64.0	36.0	64.0	36.0	64.0	36.0	64.0
37	38.0	62.0	38.0	62.0	38.0	62.0	38.0	62.0
38	40.1	59.9	40.1	59.9	40.1	59.9	40.1	59.9
39	42.3	57.7	42.3	57.7	42.3	57.7	42.3	57.7
40	44.4	55.6	44.4	55.6	44.4	55.6	44.4	55.6
41	46.7	53.3	46.7	53.3	46.7	53.3	46.7	53.3
42	49.0	51.0	49.0	51.0	49.0	51.0	49.0	51.0
43	51.4	48.6	51.4	48.6	51.4	48.6	51.4	48.6
44	53.8	46.2	53.8	46.2	53.8	46.2	53.8	46.2
45	56.3	43.7	56.3	43.7	56.3	43.7	56.3	43.7
46	58.8	41.2	58.8	41.2	58.8	41.2	58.8	41.2
47	61.4	38.6	61.4	38.6	61.4	38.6	61.4	38.6
48	64.0	36.0	64.0	36.0	64.0	36.0	64.0	36.0
49	66.7	33.3	66.7	33.3	66.7	33.3	66.7	33.0
50	69.4	30.6	69.4	30.6	69.4	30.6	69.4	30.6
51	72.3	27.7	72.3	27.7	72.3	27.7	72.3	27.7
52	75.1	24.9	75.1	24.9	75.1	24.9	75.1	24.9
53	78.0	22.0	78.0	22.0	78.0	22.0	78.0	22.0
54	81.0	19.0	81.0	19.0	81.0	19.0	81.0	19.0
55	84.0	16.0	84.0	16.0	84.0	16.0	84.0	16.0
56	87.1	12.9	87.1	12.9	87.1	12.9	87.1	12.9
57	90.3	9.7	90.3	9.7	90.3	9.7	90.3	9.7
58	93.4	6.6	93.4	6.6	93.4	6.6	93.4	6.6
59	96.7	3.3	96.7	3.3	96.7	3.3	96.7	3.3
60	100.0		100.0		100.0		100.0	

Since the only defect of the Longyear or volumetric formula in the high brackets of recovery is due to the factor "a," an empirical solution might be to substitute "i" for "a" so that as recovery increases the core analysis may be multiplied by a factor greater than from 35.5 to 51.2 pct.

The formula would then become:

$$At = \frac{S\left(100 - \frac{i^2}{100}\right) + \frac{i^2}{100}\,C}{100}$$

For brevity, we will refer to this as the i^2 formula.

By substituting zero and 100 for i, it can be seen that when core recovery is zero, the core is given no weight and when recovery is 100 pct, the weight given the core conforms with our ideal, so on two points we are immediately satisfied with the formula. There are then two avenues of approach by which we may analyze the validity of the formula; the first is by examining graphs of the different formulas and using common sense; the second is by actual application of the formula to real problems encountered in the field.

In Fig 2, the curves of three of the above formulas have been plotted on a graph whose ordinate is the percentage of weight given the core in the analysis of the whole run, and whose abscissa is the percentage of core recovery. Note that the direct proportion formula gives an exceedingly high rating to core from the outset while the i^2 curve begins its major departure from the Longyear curve in the high regions of core recovery just where we have felt dissatisfied with the volumetric figures.

Some drilling on the Vermillion Range in Minnesota would have given a particularly deceptive answer had the Longyear formula been used blindly. The material drilled was flinty, hard, magnetic, lean, unoxidized iron formation, and while the figures below represent an extreme example, it is felt that they will illustrate the point.

Depth: 590 to 595 ft.
Core recovery: 59 in. or 98.33 pct.
Core analysis: 5.98 pct Fe.
Sludge analysis: 43.72 pct Fe.
Bit size: AX.

The Longyear figures give the following:

$$At = \frac{35.4 \times 5.98 + 64.6 \times 43.72}{100}$$

$At = 30.36$ pct Fe, an analysis which might place this formation in the realm of commercial concentration by metallurgical methods. The i^2 formula gives:

$$At = \frac{96.70 \times 5.98 + 3.30 \times 43.72}{100}$$

$At = 7.23$ pct Fe, an analysis which demonstrates the worthless character of the formation.

While this case is admittedly an extreme one, an examination of many hundreds of feet of this same drilling reveals that although core recovery was mainly higher than 70 pct, the sludge ran consistently from 1½ to 3 times as high in iron as the core. There was probably some metallic iron present, caused by abrasion of the drilling apparatus in the hole by the extremely hard formation. Also concentration of magnetite by agglomeration in the sludge was noted, partly influenced by the grease used on the core barrel.

Numerous holes in active mines are being examined and comparison is being made between corresponding drill runs and mine samples to confirm or deny the validity of the i^2 formula. So far, most of the areas where comparison can be made have happened to be in low core recovery holes and here the Longyear or volumetric figures check very well. However, at no point where high core recovery was attained has the i^2 formula been found seriously in error.

As a result of this research the writer has made up a table which employs both the Longyear and the i^2 formulas. Table 1 employs the Longyear figures up to the point of coincidence of the Longyear and i^2 curves (see Fig 2) and then follows the i^2 curve to 100 pct.

Several thousand feet of drilling on several of the iron ranges in the Lake Superior District in which core and sludge have been weighted according to Table 1 have given apparently satisfactory results, and it is hoped that it may prove useful in other regions.

Coal Mine Development in Alaska

By ALBERT L. TOENGES,* Member AIME

Alaska requires an adequate fuel supply for its development, and has large potential coal reserves ranging from lignite to subbituminous and anthracite.

Coal production in the Territory now is less than the requirements. In 1947, production was 361,000 tons, divided about equally between bituminous coal from the Matanuska field and subbituminous coal from the Nenana field. There is need for development of modern mechanized mines, which should produce the required output with a minimum of workers. These mines should not be thought of in the light of large potential capacity, as in the States. However, these new mines should be developed by modern methods, which should result in lower costs than at present. Because of the diverse physical conditions in the coal fields, intensive investigation by diamond drilling is necessary to properly plan the development and equipment of mine sites at minimum cost. The Bureau of Mines is doing this now in the Wishbone Hill area of the Matanuska field. All coal land in Alaska is government-owned and subject to the Coal Leasing Act.

Coal mining in Alaska has been handicapped by an inadequate supply of dependable workers. Wage scales have been very high in recent years, and "floaters" have been attracted to the coal mines. This type of labor is not conducive to efficient operation. Modernization of mines will require skilled and dependable workers and to induce such men to become interested in coal mining in Alaska, modern facilities, such as modern houses, schools, churches, and recreation halls, must be provided near the mines. Alaska is still a frontier and improvements in living standards must accompany development of mines in order that a stable community may be established.

Coal is known to have been mined in Alaska at Port Graham, on Cook Inlet, by the Russians in 1854. During the Klondike gold rush in 1898, coal was mined in scattered areas for use on steamers plying the Yukon River and its navigable tributaries. After the Alaska Railroad was built, two important fields were developed—the Nenana and the Matanuska (Fig 1). The Bering River field was explored by the Guggenheim interests at one time, and a railroad to the area was constructed. However, the field was abandoned because of litigation and reported unsatisfactory physical condition of the beds.

There are areas that today are unimportant but which may be developed in the future. Possibly the most important of these is the deposits of subbituminous coal adjacent to Cook Inlet, near Homer, on the Kenai Peninsula (see Fig 1). It is expected that the demand for coal in the Kenai Peninsula will increase because the population in that area has increased as a result of agricultural development. Numerous outcrops of coal are exposed on the peninsula and development of a mine there would obviate dependence on supplies of coal from the Matanuska field. Coal from this field must be shipped approximately 160 miles by rail to Seward and transported thence by boat about 180 miles to Homer. There has been no regular boat service from Anchorage to Homer.

There are deposits of coal in the Arctic region but a description of them will not be given in this paper, which will describe the more important deposits in the Alaska Railroad belt. The results of a reconnaissance in the Arctic have been described.[1]

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Description of Fields

MATANUSKA FIELD

The Matanuska field lies in the valleys of the Matanuska River and its tributaries and their separating ridges. This field has been interpreted by geologists as a sunken fault block that separated the Talkeetna Range of mountains from the Chugach Range.

The Matanuska River flows west along the south side of the field and the Glenn highway, which extends from Anchorage via Palmer to the Richardson highway near Glen Allan, follows the Matanuska Valley. This road to the coal field can be traveled the year around. Although the snow-

[1] Albert L. Toenges and Theodore R. Jolley: Investigation of Coal Deposits for Local Use in the Arctic Regions of Alaska and Proposed Mine Development. U. S. Bur. Mines *R. I.* 4150 (1947) 19 pp.

FIG 1—Map of Alaska showing coal fields. (*Map courtesy U.S. Bureau ofMines.*)

fall may be 36 in., the road is kept passable.

The coal exposures occur in an area approximately 25 miles long by 7 miles wide and parallel the general trend of the Matanuska Valley. Complexity of geologic structure exists in the area and there are igneous intrusions which appear to increase progressively from west to east. These features have, no doubt, influenced the rank of the coal. This is true of the high quality of the coal found at Chickaloon on the eastern end of the area.

The coal beds, which are predominantly bituminous in rank, occur in the Chickaloon formation of Tertiary age. This formation, which is below the Eska conglomerate, comprises claystone, siltstone, sandstone, a few thin beds of fine-grained conglomerates, and coal beds. These coal beds generally occur in the upper 1400 ft of the formation. Lateral changes in the thickness of the coal beds and quality of the coal results in various coal beds being minable in one location and not in another.

Evan Jones Mine

At present, the Evan Jones mine, the only mine in operation in the Matanuska coal field (see Fig 2), at Jonesville, is on the Matanuska branch line of the Alaska Railroad about 17 miles east of Palmer by highway and 58 miles from Anchorage via Palmer. The coal ships and stores well and is preferred to the subbituminous of the Nenana field to the north. The life of this mine is limited owing to physical conditions. Most of the present production is from pillars.

The main opening of the Evan Jones mine is a single drift driven at water level. This drift passes through 300 ft of glacial gravel and approximately 400 ft of rock and then penetrates the south limb of the coal-bearing formations of the syncline, which extends for about 800 ft. Six coal beds, with dip ranging from 11 to 30° north, were found in this limb of the syncline. The main drift then extends approximately 800 ft north in rock to the coal formations in the north limb of the syncline. Ten coal beds were found in this limb, and dips range from 25 to 35° to the south.

The minable coal has been extracted from the beds in the south limb of the syncline east of the Jonesville fault, which crosses the main drift about 1400 ft north of the mine workings in this limb of the syncline.

Current production is from the No. 3 bed in the north limb of the syncline. The thickness ranges from 8 to 12 ft. A section of this bed at 8 chute main gangway, 18 crosscut, is as follows:

Description	Thickness Ft	In.
Roof—claystone	2	
Roof coal	0	5
Hard claystone	0	1
Coal	1	5
Hard claystone	0	½
Coal	1	4
Fine sandstone parting	0	½
Coal	1	1
Siltstone, thin coal stringers	0	2
Coal	0	5
Siltstone, occasional coal stringers	0	8
Coal	2	5
Hard claystone lense	0	½
Coal	0	5
Claystone lense	0	½
Coal	0	7
Bottom coaly claystone (upper contact breaks well)	1	
Bed thickness	8	1
Coal thickness	7	1

This section is typical of bed conditions and it is necessary to wash the coal to render it merchantable.

The chute-and-pillar system of mining is followed, with rooms driven up the rise of the bed from the gangway, which is driven on the strike. Rooms are turned on 50-ft centers and driven

2

3

FIG 2—Surface plant of Evan Jones mine.
FIG 3—Tipple and working Eska mine.
(*Photographs courtesy U.S. Bureau of Mines.*)

10 ft wide and about 5 ft high for a distance of about 1100 ft. A block comprises five rooms. Crosscuts between rooms are on 50-ft centers. After the room has been driven to its limit, the upper 3 to 7 ft of the bed and pillars are removed on retreat. Pillars are extracted by taking successive angle slabs off them. The coal flows by gravity to a chute on the haulageway.

The production of the mine, which averages about 450 tons per day is prepared in a surface plant. The plus 3-in. coal is hand-picked, and the minus 3-in. is washed in a jig-type washer.

Electric power for underground and surface operations is generated at the mine by a 300 kw, 440 v, 3-phase, alternating-current, steam turbine-generator.

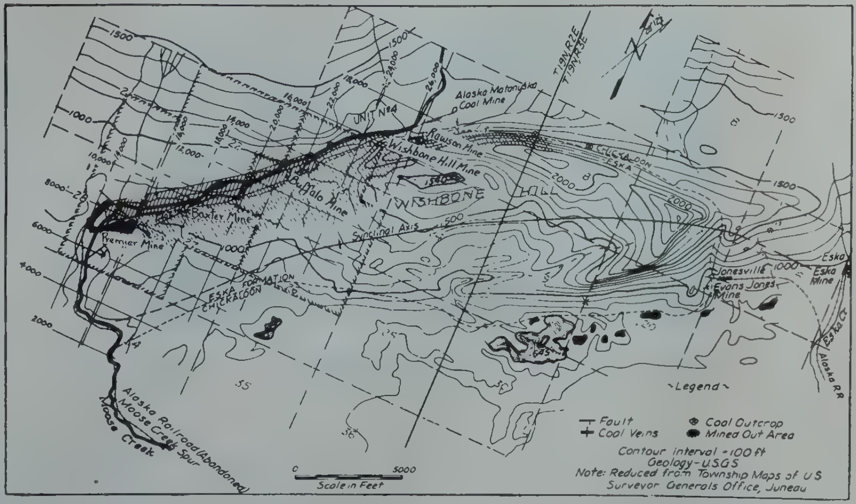

FIG 4—Map of Moose Creek District. (*Map courtesy U.S. Bureau of Mines.*)

Eska Mine

This mine (Fig 3) was operated by the Alaska Railroad but was abandoned in June 1946 owing to adverse physical and labor conditions. The mine was well-equipped with a preparation plant containing a Baum-type jig.

Moose Creek District

The Moose Creek district (Fig 4) lies on the north limb of the Wishbone Hill syncline in the western part of the Matanuska field, about 5 miles west of Jonesville. At one time a branch line of the Alaska Railroad extended 4 miles up Moose Creek to near the Premier mine. This line was abandoned after a heavy flood in 1942. The district is accessible over a 6-mile all-weather road that extends from the Glenn highway.

Coal mining first began in the Moose Creek district in 1916 and continued intermittently until after the war. The Pioneer, Baxter, Rawson, Alaska Matanuska, and Premier mines have been abandoned. Physical conditions in and surrounding the beds were the principal factors leading to their abandonment. Some areas have been disturbed by intense faulting and folding, and correlation of beds and extent are difficult to interpret from drill-hole logs.


5


6

FIG 5—Suntrana mine, Healy Coal Corp.
FIG 6—Usibelli strip mine, Healy River.
(*Photographs courtesy U.S. Bureau of Mines.*)

Buffalo Mine

This mine, which ceased operation at the close of World War II, is in Moose Creek district, 6 miles by road from the Glenn highway and 12 miles from Palmer. Coal was hauled by truck 7 miles to Moose Creek siding for shipment on the Alaska Railroad to Anchorage or directly by truck to Palmer.

The mine was developed from a water-level drift that intercepted the Nos. 2, 3, 4, and 5 beds of the Buffalo group and a slope sunk on the No. 2 bed. Mining has been conducted in the Nos. 2, 3, and 4 beds, and the main gangway has been driven in the No. 5 bed. The average thickness of the beds in the drift is: No. 2 bed, 6 ft 6 in.; No. 3 bed, 2 ft 9 in.; No. 4 bed, 2 ft 9 in.; No. 5 bed, 8 ft, with a number of shale and bone partings. The thickness of these beds varies throughout the mine, and the variations occur in a distance of about 35 ft. The dip of the beds is 65° southeast near the surface, and there appears to be a decrease in dip with depth as the basin of the syncline is approached. The interval between beds may make the mining of some beds difficult. The coals are high-volatile B bituminous and noncoking.

NENANA FIELD

This field is in the northern foothills of the Alaska Range, adjacent to the Nenana River and the Alaska Railroad. It is approximately 100 miles southwest of Fairbanks. The southern part of the area is drained by the Healy

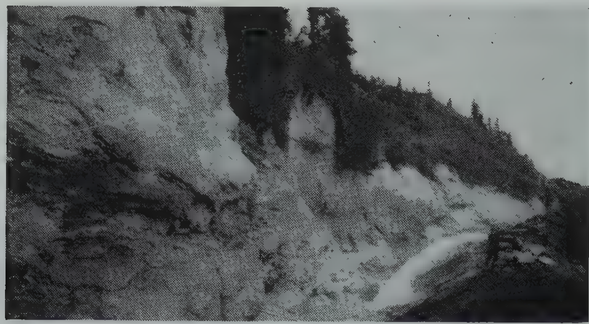

FIG 7—Hydraulicking at the Usibelli strip mine. (*Photograph courtesy U.S. Bureau of Mines.*)

River, and the northern part by Lignite Creek. Both streams are tributaries of the Nenana River.

Beds of low-rank subbituminous coal and lignite occur in the area, which is characterized by the absence of extreme structural disturbances and igneous intrusions. The thickness of the beds ranges from 6 to 50 ft and dip from 10 to 70°.

Coal has been produced from one underground mine, Suntrana, of the Healy River Coal Corp.; and the Usibelli and the Diamond strip mines.

The coal beds are exposed along the banks of Healy Creek for about 12 miles. Both banks of the stream are formed by outcrops, which are 20 to 50 ft above the bottom of the stream. The No. 1 bed is the north bank of Healy Creek for 4 miles east of Suntrana.

Suntrana Mine

The Suntrana mine (Fig 5) is on the north bank of Healy Creek. A branch line of the Alaska Railroad extends to the mine from Healy, which is approximately 112 miles south of Fairbanks and 244 miles north of Anchorage.

The main opening is a drift driven through rock at water level. This drift intersects the various coal beds that outcrop on the surface and dip approximately 18 to 28° north. The beds are numbered in ascending order from 1 to 6, and below 1 in descending order from F to A. Thickness of the beds ranges from 12 to 50 ft.

The chute-and-pillar system of mining is employed. Gangways and counters are driven on the strike of the beds, and rooms are turned on 50-ft centers and driven 8 ft wide and 8 ft high up the rise of the bed for about 200 ft. Crosscuts between rooms are on 50-ft centers. After two rooms have been developed, the pillar between the rooms is split by driving a room 8 ft wide and 8 ft high. Extraction of the full thickness of the bed begins at the top of the room by taking successive cuts from the pillar at about 45° and mining the top coal in the rooms. Batteries constructed of timber are built across the room at about 35 ft intervals. Coal is drawn into chutes from the battery until a high percentage of cave rock appears. Broken coal is drawn from rooms at a rate that will provide height for men mining the roof coal. About 2 ft of roof coal is left to support the overlying sand during mining. This top coal falls later. All coal is shot off the solid.

Coal passes by gravity in steel-lined chutes to cars on the entry. Loaded trips are transported to the tipple by electric-battery locomotives. Coal is hand-picked and screened to produce 6-in. lump, nut, and slack. The principal consuming area is Fairbanks and vicinity.

The coal is subject to spontaneous combustion, and a number of underground fires have occured which have been sealed with sand barriers.

Usibelli Strip Mine

The Usibelli strip mine is on the north bank of Healy Creek, about 2½ miles east of the Suntrana mine (Fig 6 and 7).

Production comes from the No. 1 bed, which is 35 to 40 ft thick. The strike of the bed is east-west paralleling Healy Creek. The dip ranges from 20 to 30° north. The coal bed outcrops above water level in the north bank of Healy Creek and is mined down-dip to water level. Overburden, which is an arkose, is moved by hydraulicking and with bulldozers. The coal is loaded by diesel-powered shovels into trucks and transported to the loading chute, which discharges the coal directly into railroad cars at the railroad siding adjacent to the Suntrana mine.

Diamond Strip Mine

The Diamond strip mine is about 5 miles southwest of Healy. A truck road has been constructed from the railroad siding, 1 mile north of Healy, to the mine. The road traverses glacial morains and is subject to temporary closure by drifting snows in winter.

Production comes from a 40-ft bed of coal that strikes S 62°W and dips 33° northwest with the slope of the hill. The overburden is moved with a diesel-powered shovel and bulldozer. The coal is loaded into trucks with a diesel-powered coal-loading shovel.

Future development as a strip mine is limited by the increasing thickness of the overburden and inadequate equipment.

There are other scattered coal areas near the Alaska Railroad, such as deposits near the north boundary of McKinley National Park, Broad Pass, and Colorado, but the areas that have been described present opportunities for development near rail transportation.

Summary and Conclusions

Anchorage is the largest town in Alaska, and, with Army installations, an increase in the supply of bituminous coal, which stores well, is needed. The physical conditions in the Matanuska field are such that before an investment in the development of modern mechanized mines is made, a thorough study of the area and investigation by diamond drilling is necessary. The Bureau of Mines is beginning an investigation by diamond drilling of an area in Wishbone Hill southwest of Jonesville. A study of the area by geologists of the Geological Survey indicate a large potential reserve of coal and no adverse physical conditions, such as faults of large magnitude. The objective of the Bureau's investigation is to determine minable reserves for mine sites in this area.

The development of modern mines of sufficient capacity to fulfill the requirements of Alaska should be given consideration by coal operators, but included in this development should be the establishment of living conditions that will draw to the territory the skilled workmen that will be required for these modern mines.

Electrical Dewatering of Phosphate Tailing

By E. C. HOUSTON,* Member AIME, V. J. JONES,* and R. L. POWELL†

The phosphate ores mined in middle Tennessee typically consist of granular rock phosphate particles disseminated in a clayey matrix. In the TVA plant near Columbia, Tenn., the phosphate ore is mined, made into a slurry with the addition of a small amount of sodium hydroxide as dispersant, and treated in a hydroseparator to remove minus 10 micron material. The hydroseparator underflow, comprising a rough concentrate, is transported by pipeline to the plant; the hydroseparator overflow, comprising a tailing, is flocculated by addition of calcium sulphate and is discharged to settling ponds at a rate of about 1400 gpm. Sedimentation in the ponds produces a clarified effluent, which may be recycled for use as process water or discharged to surface drainage. This method of tailing treatment is not entirely satisfactory since poor sedimentation characteristics of the tailing result in poor ultimate utilization of pond storage volume. The sediment contains about 70 pct water, even after several years of settling, and is not sufficiently dry to permit it to be used as back-fill, which would provide a method for reclaiming ponds that had been filled with sediment.

Since important advantages would result from a process whereby the water content of the tailing solids would be made substantially lower than that obtained in the present process, several possibilities of achieving this were considered. Flocculation tests were made with a wide variety of chemicals, but none gave any improvement over the results obtained with calcium sulphate. Filtering or centrifuging was found to be infeasible. Small-scale tests of dewatering by electrophoresis showed promising results, so this method was studied further in pilot-plant equipment. It was found that the tailing could be dewatered by electrophoresis to produce the solids in a form suitable for use as back-fill. However, the dewatered tailing has no monetary value at present, and the process is not economically competitive with settling pond operation under present conditions.

This paper describes the pilot-plant work on electrical dewatering. It is believed that the data may find application in dewatering physically similar but economically valuable materials such as clays or mill slimes, or in dewatering phosphate tailing in the event of increased settling pond costs

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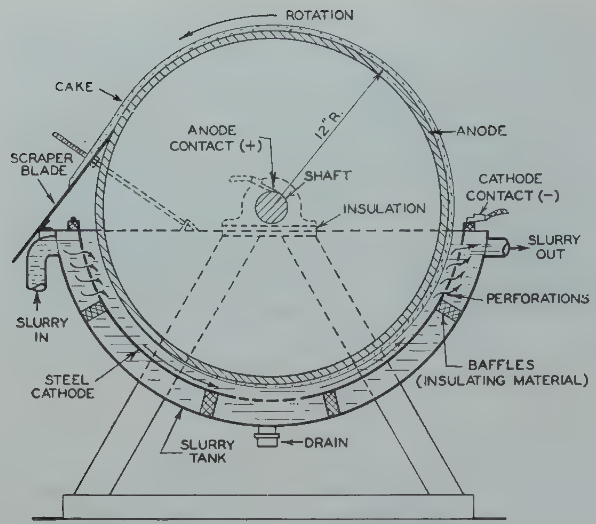

FIG 1—Experimental machine for electrical dewatering of tailing: cross section through center.

or the development of end uses for the tailing solids.

Initial Considerations

Electrophoresis may be defined as the migration of particles in liquid suspension resulting from the application of direct current electricity to the suspension.[1,7] This phenomenon has been found to be applicable to the separation of particles from suspension by deposition on an electrode immersed in the suspension. Although the principles of electrophoresis have been the subject of numerous laboratory investigations,[1,4,7] there have been few commercial applications of the phenomenon to the separation of solids from liquids. This, according to Creighton,[3] is probably due in part to lack of knowledge of conditions necessary for best results. It is also due to the availability of simpler and better-known methods that will usually accomplish solids separation at lower cost, namely, settling, filtering, or centrifuging.

In work on the electrical dewatering* of clays, Speil and Thompson[7] found the following conditions to give the best results: high solids concentration in slurry, low conductivity of slurry, high temperature, close spacing of electrodes, and effective mechanical circulation to bring the suspended particles close to the anode. The kind and dosage of dispersant chemical were shown to have important effects that depended on the particular clay being treated. In the present study, an effort was made to fix these variables as near to expected optima as possible prior to starting experimental work. Since, at the TVA plant, it would be feasible to operate with a hydroseparator overflow (dewatering machine feed) containing from 12 to 14 pct solids, 12.5 pct was selected for use in the experimental work. Calculations showed that if the plant were operated with this solids concentration in the hydroseparator overflow it would be necessary that the dewatering machine remove about 35 pct of the solids from the slurry in order that the required rate of solids disposal be achieved. The dewatering machine effluent would contain about 8 pct solids and would be recycled to the mine for use in slurrying fresh ore. Conductivity of the slurry would be governed by suspensoid concentration (fixed as in the foregoing), on kind and dosage of dispersant chemical, and on naturally occurring electrolytes in the process water which is taken from the Duck River. In view of the relatively large volumes of slurry to be handled and the lack of a low-cost source of heat at the plant, preheating of the slurry prior to electrophoresis was considered impractical.

Those variables that would not be controllable in plant-scale operations, such as temperature and natural impurities in the process water, were adjusted in the experimental work to simulate average plant conditions. The investigation included study of the effects of deposition time, type of dispersant chemical, current density, and slurry concentration; the possibility of selective size separation was investigated, and tests were made of several anode materials.

[1] References are at the end of the paper.

* The term "electrical dewatering" as used in this paper denotes electrophoretic separation of solids from aqueous suspension.

Electrical Dewatering Tests

EQUIPMENT AND MATERIALS

Equipment design, materials, and procedures used in the experimental work were selected with a view to applicability to plant-scale operations. Of the several types of electrophoresis machines proposed in the literature,[4] two appeared to be promising for this specific application. These were (1) the rotating drum type and (2) the continuous-belt type. The rotating drum machine is similar in appearance to a rotary vacuum filter except that the drum has a solid conducting surface that serves as the anode; the solids adhere to this drum and are scraped off with a fixed blade. The continuous-belt type machine has for an anode either an endless metallic strip partly immersed in the slurry or a stationary metal plate over which is drawn an endless cloth strip. The first type has been used for commercial dewatering of kaolin;[4] there are no reported commercial applications of the second. In the present work, small-scale preliminary tests were made in a machine of the continuous cloth belt type in which a 6 in. wide canvas strip was drawn across the inclined steel bottom of a shallow rectangular tank, which contained the tailing slurry. The bottom of the tank served as the anode. The cathode consisted of a flat steel plate, which was suspended ½ in. above and parallel to the anode. It was found that deposition of solids occurred both on the surface of the belt and between the belt and the anode, which made this type of machine unsuitable. Therefore, it was decided that the rotary drum type of machine was the more promising for the intended use.

An experimental electrophoresis machine for the dewatering tests was constructed based on Schwerin's idea;[2] a cross-sectional sketch of the machine is shown in Fig 1. The principal parts were: a semicylindrical tank to hold the slurry, a semicylindrical sheet-steel cathode suspended in the tank, and, concentric with this, a revolving drum of electrically conductive material that dipped into the liquid and served

as the anode. A fixed scraper blade was provided to remove deposited solids from the surface of the rotating drum; this was set so that it would just clear the drum surfaces without effecting metal-to-metal contact. The machine differed from Schwerin's design principally with regard to the path of slurry travel; instead of feeding into the bottom of a relatively deep slurry tank provided with agitators and directing the flow upward through perforations in the cathode, the flow was directed substantially parallel to the surface of the anode in the electrode region. This reduced the dead space to only about 2 in. between the cathode and the tank; baffles were provided to prevent short-circuiting of the slurry through this region. Both the anode and the cathode were electrically insulated from the slurry tank by means of fiber sheets and strips, and the ends of the drum were painted with insulating varnish to prevent deposition on those areas. The slurry feed and discharge lines were provided with sections of Saran to reduce current leakage through the piping. The anode was 2 ft in diameter by 2 ft long. In most of the tests the anode consisted of a section cut from gray cast-iron pipe and closed by circular steel plates at the ends; some tests, however, were made in which the anode was material other than cast iron, as will be discussed later. The anode was given a rough-machined surface prior to installation. It was rotated by means of a chain-and-sprocket drive from a variable speed reducer, which was driven by a ½ hp motor. The electrode spacing was ⅜ in. in most of the tests although a ⅝-in. spacing also was used. Although a spacing smaller than ⅜ in. could have been used, this was thought to be about the minimum feasible for use in a larger, plant-scale machine without encountering mechanical difficulties. The depth of immersion of the anode cylinder was about 9 in., and the submerged area was about 5.7 sq ft; this area was used in calculating anodic current densities and deposition rates. Electric power was supplied from an 8 kw, dc motor generator, with contact to the anode by means of a brush; power was varied by means of a resistor and was measured with an ammeter and voltmeter. The voltage range was 0 to 100. A view of the machine in operation is shown in Fig 2.

FIG 2—Laboratory machine for electrical dewatering of phosphate tailing.

Since the experimental work was not carried out at the Columbia, Tenn., plant, but at the Wilson Dam, Ala., laboratory of TVA, it was not convenient to use tailing slurry as produced in the hydroseparator. Test slurries for experimental work were prepared in the laboratory from tailing sediment dug from a settling pond at the plant and shipped in moist condition to Wilson Dam. Although the composition of the tailing varied somewhat, the following analysis represents a typical sample in per cent by weight on dry basis: P_2O_5, 9.0; CaO, 10.0; SiO_2, 37.4; Fe_2O_3, 7.9; Al_2O_3, 18.2; loss on ignition, 11.5. All of the particles were smaller than 10 microns and more than 70 pct were smaller than 2 microns. The finer fraction exhibited typical colloidal characteristics, namely, high base exchange capacity and high degree of hydration. In preparing slurries for test purposes the solids were dispersed mechanically in tap water in a tank by means of propeller mixers, and the slurry was adjusted to the desired concentration and dispersant dosage. The laboratory tap water was similar in total solids content to the plant process water. Test slurries were pumped to a constant-head tank that fed the dewatering machine at controllable rates up to 5 gpm. The capacity of the slurry preparation equipment was about 1000 gal, which was sufficient for several hours of continuous operation.

EFFECT OF DEPOSITION TIME

In the electrical dewatering of clays, Speil and Thompson[7] found that the optimum deposition time was that which allowed sufficient dewatering of the deposit without increasing the resistance of the deposit to such a point that the energy consumption per unit of clay increased noticeably. The optimum deposition time varied with different clays. In the present study a series of tests were run in the experimental dewatering machine to determine the effect of deposition time. For these tests, an electrode spacing of ⅝ in. was used so that deposition time, hence cake thickness, could be varied over a wider range than would be permitted by the ⅜ in. spacing that was considered to be the minimum practical for large-scale use. Although the wider spacing caused high power consumptions, the tests served to indicate the relative effects of deposition time on power consumption and on other process variables. The current density was held at 9 amperes per square foot; feed slurry concentration was 12.5 pct; the duration of each test was 30 min; and sodium hydroxide was used as the dispersant at a dosage of about 4 lb per ton of solids, which is the dosage generally used in the plant. The deposition time was varied over the range 0.5 to 6.0 min by varying the speed of rotation of the drum from 4.2 to 60 rph.

The cake thicknesses obtained varied from about ⅛ to ½ in., and the moisture in the cakes was 52 to 58 pct. Results of the tests (Fig 3) indicated that increases in the deposition time served to decrease the power requirement and the deposition rate by only small and approximately equivalent percentages. It was necessary to decrease the voltage with increase in deposition time to maintain a constant current input; this indicated that the deposited material was more conductive than the slurry. It was concluded, therefore, that, in the range of deposition time studied, resistance of

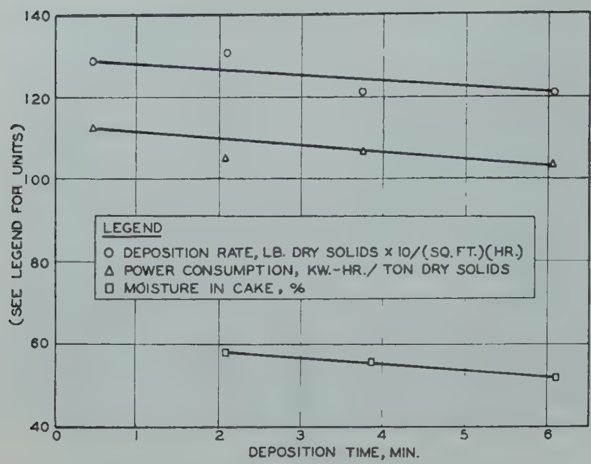

FIG 3—Effect of deposition time in electrical dewatering of phosphate tailing.

Constants: feed concentration, 12.5 pct; dispersant, NaOH (approximately 4 lb per ton solids); solids removed, 43.5 pct of feed solids; current density, 9.0 amp per sq ft; electrode spacing, ⅝ in.

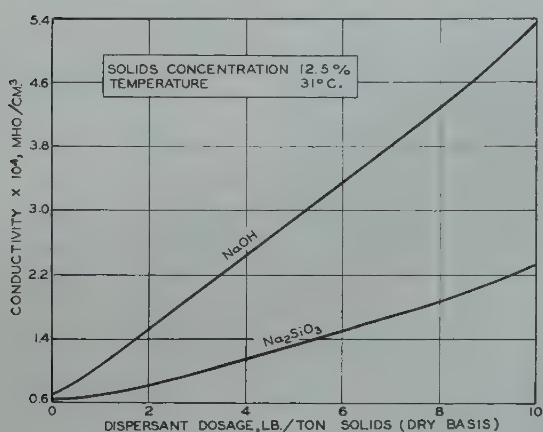

FIG 4—Effects of NaOH and Na_2SiO_3 dispersants on conductivity of tailing slurry.

the cake would not be a limiting factor in machine design and operation. The deposition time should be adjusted to suit the electrode spacing; close spacing would require shorter deposition time. The optimum deposition time would be that which would result in a cake of maximum thickness consistent with providing adequate clearance for the flow of slurry between the cake and the cathode. The optimum electrode spacing would be the minimum consistent with provisions for a reasonable cake thickness and for flow of slurry between the electrodes; as previously stated, the minimum practical spacing was believed to be about ⅜ in. A decrease in power requirement with increase in deposition time was observed up to and including 6 min, which was the maximum used in these tests. Speil and Thompson[7] found, however, that an increase occurred after 10 to 12 min, so it is probable that in the present work an increase would have occurred with longer deposition times. It is believed that the relative effects of cake thickness would be substantially the same with sodium metasilicate, which, in later tests, was found preferable to sodium hydroxide as a dispersant.

EFFECT OF DISPERSANT CHEMICAL

With regard to the best kind and amount of dispersant chemical, preliminary tests showed that sodium hydroxide in a dosage of about 4 lb per ton of tailing solids or, as an alternative, sodium metasilicate at 5.2 lb per ton would be equally suitable from the standpoint of obtaining satisfactory dispersion of the phosphate pulp. Laboratory tests with several other chemicals showed that, although satisfactory dispersion could be obtained with some, the costs for the required dosages would be considerably greater than those for sodium hydroxide or sodium metasilicate. The study of effect of dispersant on electrical dewatering was confined, therefore, to a consideration of these two chemicals, and conductometric titrations were made to determine their relative effects on slurry conductivity since electrophoretic efficiency in dispersed slurries was known to be inversely proportional to conductivity.[1] The slurry samples used were prepared by mechanically dispersing tailing solids in distilled water, and the solids concentrations were adjusted to 12.5 pct by dilution. Sodium hydroxide or sodium metasilicate in the form of 2.5 pct solutions of reagent-grade chemicals was added to the slurry in increments from a burette, and conductivities were measured 20 min after each addition by an alternating current comparison bridge using a dip cell with black platinized electrodes. The reagent dosages were carried somewhat beyond the amounts required for adequate dispersion to determine the effects of overdosage. The results of the titrations (Fig 4) showed that dispersion with sodium metasilicate increased the conductivity considerably less than did dispersion with sodium hydroxide, and it was concluded that sodium metasilicate would be the more desirable dispersant for use with electrical dewatering. It was also concluded (Fig 4) that overdosage of either dispersant would increase the power consumption, although this effect should be less pronounced in the case of sodium metasilicate.

EFFECT OF CURRENT DENSITY

A series of tests were run in the dewatering machine to determine the effects of anodic current density. Since increase in current density would be expected to increase the deposition rate, the feed rates were increased with increase in current density in an attempt to cause a constant percentage of the solids to be removed from the slurry throughout the series and thereby maintain the same average solids concentration in the slurry in the machine. If the feed rates were not adjusted in this way, the power requirement at high current densities would be too high since the more rapid deple-

tion of the solids concentration of the slurry would result in a lower average solids concentration in the machine.

The tests showed (Fig 5) that with an increase in anodic current density within the range 3.7 to 16 amp per sq ft the power requirement increased from 35 to 125 kw-hr per ton of solids removed and the deposition rate increased from 4.5 to 16 lb of solids per square foot per hour; both effects were apparently linear throughout the range investigated. With an increase in current density, the moisture content of the dewatered solids decreased progressively from 62 pct at 3.7 amp per sq ft to 50 pct at 16 amp per sq ft. Because of inability to adjust the feed rate with sufficient accuracy to give a constant percentage removal of solids for all current densities, the average removal was about 40 pct instead of the 35 pct sought. Deviations in percentage removal of solids was believed to be largely responsible for the scattering of data in Fig 5. Although the tests were made with an electrode spacing of ⅜ in., the correlation of deposition rate and cake moisture with current density should be directly applicable to other electrode spacings. The power requirement, however, would increase with increase in electrode spacing.

Selection of current density depends on an economic balance between the cost of power and the equipment cost for a particular dewatering operation. With high current density the power requirement would be relatively high, but the required anode area and hence the equipment requirement would be relatively small. An estimate prepared for the TVA plant showed that a current density of about 7.4 amp per sq ft should be close to the value most economical for that particular installation; it is evident that the value would be different for a different set of conditions.

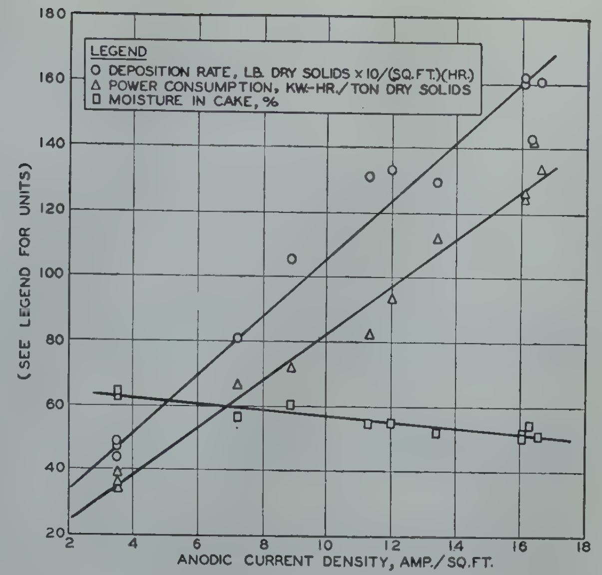

FIG 5—Effect of current density in electrical dewatering of phosphate tailing.

Constants: feed concentration, 12.5 pct; dispersant, Na_2SiO_3 (approximately 5.2 lb per ton solids); solids removed, 40 pct of feed solids; electrode spacing, ⅜ in.

EFFECT OF CONCENTRATION OF SLURRY

Although Speil and Thompson[7] had demonstrated the beneficial effects of using a high concentration of solids in electrical dewatering of clays, it was desired to obtain quantitative data on this effect with phosphate tailing. A series of three tests were run in the dewatering machine to determine the effect of concentration of slurry. The feed rate was adjusted in each test to supply solids to the machine at the same rate although the volume rates increased with decrease in concentration. The effect of slurry concentration within the range 7.5 to 12.0 pct solids is shown in Table 1.

Table 1 . . . Effect of Slurry Concentration

Feed Slurry Concentration, Per Cent Solids by Weight	Power Required, Kw-hr per Ton Dry Solids Deposited	Deposition Rate, Lb Dry Solids per Sq Ft per Hr
12.0	82	12.7
8.7	126	9.2
7.5	157	7.9

The data in Table 1 served to emphasize the importance of a high solids concentration in the feed slurry and confirmed the selection of as high a concentration as was commensurate with efficient phosphate plant operation.

SIZE DISTRIBUTION OF PRODUCTS

The applicability of electrical dewatering would be dependent on recycling all of the depleted effluent from the dewatering machine to the slurrying operation, since the effluent would contain too high a concentration of solids to permit it to be discharged to surface drainage. Therefore, it was important to determine whether a selective particle-size separation occurred, since such action would result in an accumulation of fine or coarse particles that might make the process inoperative. Five sets of samples of feed, cake, and effluent from laboratory dewatering tests were analyzed for particle-size distribution by the pipette method.[6] The results, which are summarized in Table 2, indicated that size segregation, if any occurred, would be too small to cause adverse effects. It is probable, however, that tailing containing solids much larger than 10 microns would require a removal of coarse particles prior to electrical dewatering, since the process is theoretically not applicable to coarse materials.

Table 2 . . . Particle-size Distribution in Feed and in Products of Electrical Dewatering[a]

Material	Per Cent of Total Solids, Dry Basis		
	+10 Micron	−10 + 2 Micron	−2 Micron
Feed	1.2	19.9	78.9
Cake	1.6	19.7	78.7
Effluent	0.4	21.6	78.0

[a] Average of determinations made on five sets of samples.

TESTS OF VARIOUS ANODE MATERIALS

Characteristics of the tailing slurry were such that chemical corrosion of equipment would not be expected to constitute a major problem; however, electrolytic corrosion of the anodes would be expected to be appreciable and therefore would require consideration in selecting an anode material. Since contamination of the tailing with corrosion products would not be detrimental, the principal factors in the selection of the most suitable anode material would be: (1) cost versus

corrosion rate, (2) structural suitability and electrical conductivity, and (3) characteristics of the products of corrosion, namely, adherence to the anode and electrical conductivity.

If all of the current flowing through an anode is active in the electrolytic corrosion of metal, with no evolution of free oxygen, the weight of metal corroded, W, may be obtained by Faraday's law: $W = itK$, where i is current, t is time, and K is the electrochemical equivalent of the metal.[5] A consideration of the electrochemical equivalents of the more common conductors and their cost and structural suitability for the intended use led to the selection of the following materials for testing: iron, graphite, and stainless steel. Lead was also tested because its use has been reported in commercial electrical dewatering of kaolin.[4] The use of one of the "electrically conductive" rubber compounds as an anode coating was considered but was rejected as unpromising because of the relatively high resistivity of commercially available materials of this class. The usual methods for making chemical corrosion rate tests on small specimens were not applicable to the testing of electrophoretic corrosion, principally because of lack of a convenient method for the continuous removal of the deposited solids from the specimens while maintaining relatively constant conductivity, solids concentration, and dispersant dosage in the slurry. The only test procedures that appeared feasible were: (1) the use of a small-scale model of the laboratory machine or (2) the use of the laboratory machine as a corrosion test unit. The first method would have the advantage of using anodes small enough to weigh with sufficient accuracy to estimate corrosion by weight loss, although it would have the disadvantage of requiring the construction of a replica of equipment already on hand and would probably present difficulties in control of operation. It was decided that neither method would be likely to give dependable quantitative data on the subject but that tests made with the laboratory machine should provide qualitative information sufficient for determining the relative suitability of the four materials under consideration. Anodes to fit the laboratory machine were made from furnace-electrode graphite, gray cast-iron pipe, chemical lead sheet, and Type 430 stainless steel sheet, respectively. Each of the anodes was tested by operation in the machine for periods of at least 20 hr under the following dewatering conditions: slurry concentration, 12.5 pct; dispersant, sodium metasilicate; current density, 12 amp per sq ft.

Passivity, which would have been shown by evolution of oxygen bubbles, was substantially absent in all the tests. The graphite anode showed a definite softening after 20 hr, which indicated that it would not be suitable; the cast-iron anode showed a relatively uniform pattern of corrosion, with some pitting; and the lead and the stainless steel anodes formed hard coatings which broke off periodically in patches and resulted in a nonuniform type of wear. Both the lead and the stainless steel required several hours of operation in the machine before satisfactory adherence of cake was obtained, although satisfactory cake adherence was obtained from the start with the graphite and cast-iron anodes. Because of the uniform type of corrosion found for cast iron, together with its relatively low cost and structural suitability, this material appeared to be the most satisfactory of those tested. Since practically no evolution of oxygen gas was noted, and in view of the lack of quantitative data on corrosion rate, it was assumed that the corrosion rate would correspond to 100 pct current efficiency; namely, the weight loss of the anode would be represented by the Faraday equation, $W = itK$. Whether this assumption is conservative or optimistic is open to question; a long-term test would be required for reliable data. If the electrochemical equivalent of cast iron is assumed to be the same as that of pure iron, which for present purposes was considered sufficiently accurate, the corrosion rate of cast-iron anodes operated at a current density of 7.4 amp per sq ft would amount to about 0.25 lb of cast iron per ton of tailings dewatered. The cost of anode replacement would enter into the economic balance used to determine optimum current density.

Summary and Conclusions

Phosphate tailing, consisting of a 12.5 pct slurry of minus 10-micron fraction of Tennessee brown phosphate ore, can be dewatered by electrophoresis to 50 to 55 pct water content. A rotating drum type of continuous dewatering machine with a cast-iron anode is suitable for the process.

For best results, the slurry fed to the dewatering machine should contain a maximum percentage of solids commensurate with efficient washing plant operation. The most economical operation is obtained when the solids are removed only to the degree that would permit recycling of the effluent as process water. Sodium metasilicate is preferable to sodium hydroxide as dispersant since the former imparts a lower electrical conductivity to the slurry. The optimum anodic current density depends on an economic balance between power costs and depreciation costs since increase in current densities increases the power requirement per unit of solids dewatered and increases the deposition rate per unit of anode area. For the economic optimum under the conditions of the TVA application, namely, 7.4 amp per sq ft, the power requirement would amount to about 65 kw-hr per ton of solids removed, for which the corresponding deposition rate would be about 8 lb per sq ft per hr.

The study indicated that under present conditions the process is not economically competitive with the use of settling ponds. Electrical dewatering, however, permits recovery of the tailing in a relatively concentrated form potentially suitable for further processing, for example, in the manufacture of fertilizers, or heavy clay products. It is hoped that the process will stimulate the development of uses for this dewatered tailing, which would improve the economics of the process.

Acknowledgments

The authors wish to express their appreciation to A. B. Phillips for design of experimental equipment and to T. P. Hignett for helpful suggestions and advice.

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Occurrence and Exploration of Barite Deposits at Cartersville, Georgia

By THOMAS L. KESLER,* Member AIME

Introduction

Essentially all of the barite produced in Georgia has come from the Cartersville district in the northwest part of the state. The earliest recorded shipment of ore, 60 tons, was made in 1894.[1] With the exception of the four-year period 1931 to 1934, the separate yearly output has been recorded and published since 1915, whereas the total yearly output of the United States has been published since 1880. The separately recorded production of Georgia (Cartersville) through 1947 is 2,232,544 short tons valued at $14,900,746,[2] but unpublished data on the period 1931 to 1934, plus the estimated unrecorded output prior to 1915, make the actual total about 2,400,000 short tons. This is about 21 pct of the total production of the United States, which, through 1947, is 11,328,440 short tons valued at $71,683,386.[2]

The barite-producing part of the Cartersville district is relatively small, having from north to south a length of 4.5 miles, and a width of 2 miles. This small area, containing 35 barite mines, is the source of practically all of the barite produced in the state. The area is hilly, and has a total relief of 500 ft. The west-flowing Etowah River crosses the middle of the area, and is an unfailing source of water for operations on properties near its course. Opencut mining is carried on during the entire year, but is hampered by rain in the winter and early spring. Three railroads, two U.S. highways, and a network of graded roads provide access to and within the district, as well as excellent shipping facilities for ore and supplies. In addition to the barite, the district contains deposits of manganese, brown iron, ocher, umber, and specular hematite.

Geologic Setting[3]

The area containing the barite deposits is underlain by Lower Cambrian rocks and their weathered residua. These rocks are grouped in three formations; in ascending order, these are the Weisner, the Shady, and the Rome.

The Weisner formation consists mainly of finely micaceous metashale containing many random beds of quartzite, a few beds of conglomerate and metasiltstone, and beds of crystalline limestone that is apparently scarce. The formation is more than 1000 ft thick, and the base is not exposed. On account of their resistance to weathering, the rocks of the Weisner formation have sustained the higher elevations and therefore crop out on the ridges that characterize the area. Nearly all of these ridges are underlain by asymmetric anticlinal folds, and conversely the intervening valleys are underlain by synclinal folds. A diagrammatic section, showing typical mode of exposure of the Weisner rocks and their relation to overlying formations, in the limb of a fold, is shown in Fig 1.

The Shady formation here consists of a lithologically distinctive series of variably siliceous specular hematite beds and thin beds of dolomite, which in places contain abundant fossils. In the zone of weathering, the dolomite has been leached and most of the hematite hydrated to ocherous and umberous clays, but bedding planes in these weathered materials are commonly well preserved with or without distortion effected by slumping. This preservation of structure during weathering is lacking in the residuum that rests on the Rome dolomite, which is described below. The maximum thickness of the

St. Louis Meeting, October 1948.
TP 2627 H. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Nov. 30, 1949. Manuscript received Jan. 12, 1949.

* Geologist, Thompson, Weinman and Co., Cartersville, Ga.

[1] References are at the end of the paper.

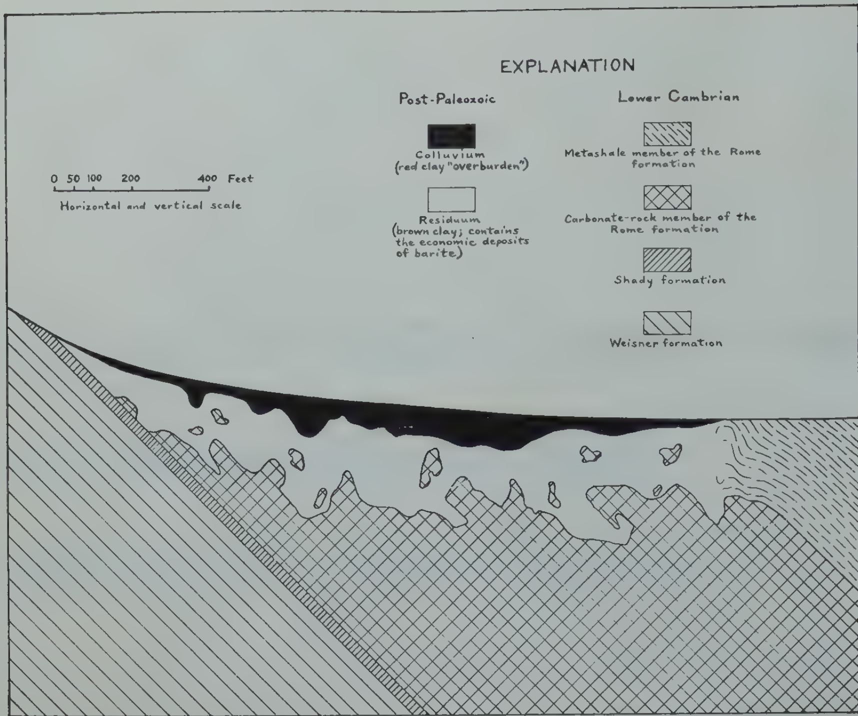

FIG 1—Diagrammatic cross section showing typical association of geologic formations, residuum, and colluvium, and their relation to the land surface. The dip is to the east.

Shady appears to be about 30 ft, and its beds are conformable with, but not everywhere present above, those of the Weisner. The Shady apparently is broadly lenticular, and where it is absent the Weisner is overlain by the Rome formation. The Shady formation has not been recognized previously as here defined, the name having been applied heretofore to the dolomite now included in the Rome formation. For a more complete discussion the reader is referred to the report[3] cited at the beginning of this section.

The Rome formation includes two members, one consisting of crystalline carbonate rocks and the other of metashale. The carbonate-rock member is the more important in the area that contains the barite mines, and there consists largely of dolomite 500 to 1200 ft in stratigraphic thickness. The dolomite is seen only in rare pinnacles and residual boulders exposed in mining ores from a deep mantle of residual clay. This clay is derived in small part from the insoluble constituents of the dolomite, and in large part from constituents of formerly overlying shaly rocks; and all of the constituents have been mixed through long-continued slumping into growing caverns and sinkholes, a process that is still active. Lateral collapse of the Rome metashale into the residual clay is indicated in Fig 1. The clay underlies the slopes of the ridges and parts of the intervening valleys, and is light to dark brown in color. It is known to be more than 100 ft thick in many places, and more than 200 ft thick in a few, and it contains the commercial deposits of barite.

The barite now in the clay was freed from the dolomite by ground-water leaching, and the deposits of economic importance are therefore of residual origin. Originally, the barite and other ore and gangue minerals were deposited in and near faults in the dolomite near the close of folding in late Carboniferous time. At that time, pronounced differences in the lithology of the different formations in the adjacent region caused unequal compression of folds, and therefore unequal shortening. The result was pronounced curvature of the regional strike, in the area now containing the barite deposits, with the development of rotational stress and its consequent pattern of ruptures, as demonstrated in the experiment of Mead.[4]

These ruptures are the faults with which the barite deposits are associated, and they occur in three differently oriented groups. One group trends parallel to the prevailing strike, which, owing to the curvature already mentioned, is N25°E to N25°W. Another group is oriented northwest of the prevailing strike, averaging about N65°W. The third group is oriented northeast of the prevailing strike, averaging about N55°E. Parts of the dolomite, and vein carbonate in the fault zones, were replaced by quartz following the deposition of the ore minerals. The jasperoid thus formed commonly contains brecciated barite, showing that fault movements continued after deposition of the ore minerals.

The major problem involved in the search for new ore is the discovery and tracing of the barite-bearing fault zones. Most of them appear to be steeply inclined, but, owing to the strong effects of slumping and erosion, the direction and amount of displacement are generally obscure. Most of the evidence of these structures must be obtained, and their trends projected, from the ridge crests where beds of the Weisner formation crop out, and are offset and brecciated along the faults. On the slopes, the residual clay that overlies the Rome dolomite is covered by red colluvial clay composed of the coarser materials formerly eroded from rocks that cropped out upslope. Even the gullies, therefore, do not expose the underlying brown residual clay except on the upper slopes where the colluvium is thin. On the lower slopes, the colluvium is commonly 20 to 50 ft thick, and, exceptionally, 100 ft thick where it appears to have filled deep sinkholes.

The Barite Deposits

The residual clay that forms the matrix of the barite ranges in color from yellowish to chocolate brown in contrast with the overlying red colluvial clay. The brown clay is quite tough and dense, averaging about 140 lb per cu ft in place and undisturbed. As its colors suggest, the clay is variably ferruginous. The term residuum is applied to the clay with its content of harder fragmental materials.

The most common of the fragmental materials is jasperoid, which occurs as angular boulders of fine-grained quartz rock stained yellow by ferric hydroxide. Thin sections of the rock show preserved carbonate texture and cleavage, as well as minute residual grains of dolomite enclosed in grains of the quartz. The jasperoid, like the barite, was freed through solution of the dolomite.

The residuum also includes random boulders of quartzite from the Weisner formation, which crops out at higher elevation. These boulders were trapped in early sinkholes, and were mixed with the residual materials by slumping as

the chemical weathering of the dolomite progressed. Similarly, flakes and slabs of partly hydrated specular hematite from the Shady formation have become mixed into the residuum, and are a source of iron contamination in the barite ore.

The residual barite is identical in all respects with barite that is exposed in a few places in uneven veins in the dolomite. It occurs as fragments of irregular shape and size, ranging in maximum dimension from a fraction of an inch to 4 ft, but usually less than 6 in. Each fragment is an aggregate of coarse crystals without crystal faces; the cleavage is pronounced and curved, and the aggregate is snow white to bluish white although thin plates are colorless and transparent. The barite contains very small amounts of quartz and of sulphides that are largely destroyed by weathering. These include pyrite, galena, sphalerite, chalcopyrite, and tennantite, and all of them except pyrite occur so sparsely and in such fine grains that they can be identified only with the microscope. The only product of weathering of the sulphides that is persistent and common enough to be a variably serious impurity of the barite ore is limonite. It occurs attached to the barite and as separate fragments sometimes a foot or more across. Their source is shown by masses of pyrite, only partly weathered to limonite, that have been found in the deeper parts of some of the mines.

The proportion of barite in the residuum is highly uneven. There is no established minimum average grade for profitable operation, owing to considerable differences in accessibility of deposits, amount of necessary dead work, length of haul, and method of concentration. Five mines of moderate to large size previously have shown overall concentration ratios, including both mining and incidental stripping, of 2.9 to 4.6 cu yd of clays in place per long ton of concentrates. These ratios probably no longer can be equalled on a large scale owing to the depletion of ore with shallow overburden. In fact, relatively good bank ore alone, without including material stripped, now shows a concentration ratio of five or more cubic yards per long ton of concentrates.

General Difficulties of Prospecting

Owing to the increasing depth of mining, the circular, hand-dug test shafts so common in earlier years, are no longer sunk in the search for new ore. Few of these exceeded 30 ft in depth, but they were adequate as a guide to mining in areas where the average overburden was thin, as in most of the Paga No. 1 mine shown in Fig 2. This mine which produced about 635,000 long tons of concentrates from 2,250,000 cu yd of residual and colluvial clays, reached a maximum depth of 150 ft. Such depth of ore-bearing residuum is an incentive for deep testing and low-cost stripping in ore-bearing areas with thicker overburden. Deep testing requires mechanical drilling.

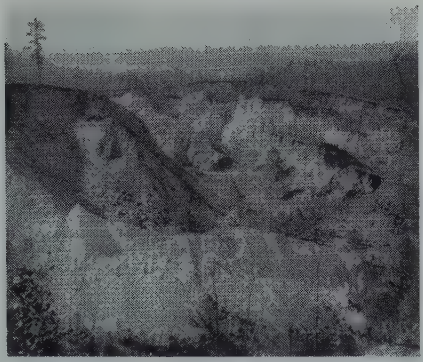

FIG 2—Paga No. 1 barite mine with maximum depth of 150 ft.

Both the colluvial and residual clays as well as the barite may be cut easily with any type of drilling bit, and the walls of drill holes in the clays stand well without casing. These features would permit rapid and inexpensive prospect drilling if the clays were free from boulders of hard jasperoid and quartzite. Owing to the presence of these boulders, it is unusually difficult to obtain satisfactory drill samples of the bank ore to desired depths at systematic, predetermined locations.

Drive-pipe and coring methods, which are commonly used to obtain samples of undisturbed bank ore, are effective only to the random depth at which a hard boulder is encountered. As the clays do not hold the boulders rigidly enough for the use of the diamond or rock bit, such encounters would mean the loss of many holes in any deposit, and of most holes in some, at depths much too shallow. Coring with a plain-end single-tube barrel has given poor recovery because the dense clay and the larger fragments of barite tended to plug the barrel. The end of the barrel was flared even by rotten boulders, and had to be sawed off frequently; hence the use of a plain end. Encounters with these boulders required the use of chopping bits with attendant loss of core.

Power-auger and churn-drill methods, which are commonly used to obtain cuttings representing complete bank ore, involve almost equally serious difficulties. The power auger not only would be stopped by hard boulders, but also would receive a high rate of wear with rapid reduction of gauge. The churn drill probably would penetrate the residuum to any desired depth, but it is doubtful whether its samples would yield sufficient data on the physical properties of the bank ore. The writer has had no opportunity to test this conclusion, but it seems likely that the soft barite would be so finely crushed that the approximate proportions of originally attached impurities could not be determined. It is important, for instance, to know the amount of iron bound to the barite, not the total amount of iron in a sludge in large part consisting of ferruginous clay.

Drilling and Sampling

Complete bank-ore samples, whether cored or bailed, are not essential for estimating ore reserves in clay-matrix deposits such as these. The first product in milling this type of ore is a log-washer concentrate, which includes the barite with attached and unattached impurities on which the balance of the milling is focused. In exploratory drilling in the interest of the Paga Mining Co., the writer obtains, through the action of the drill and supplemental washing, samples fairly equivalent to log-washer concentrates, thus bypassing the difficulty of obtaining proper bank-ore samples. A stage has not been reached yet at which estimates can be checked against subsequent mining yields, and this may require a long period of time. In the meantime, others with similar problems may find the method useful, with improvement or change as conditions may require. The various stages of the process are illustrated in Fig 3 to 6.

A company-owned "seismograph" drill is used which is equipped with a 4 by 5 mud pump and is mounted on a ton-and-a-half truck. It is driven by the truck engine. All barite prospecting is done with N-rods and 3½ in. pilot-type three-wing bits. The pilot bit is superior to other types thus far used in cutting weathered boulders and in maintaining a reasonably straight hole. It will not cut fresh boulders, however, and a few holes accordingly are lost at insufficient depth. Most of the holes range from 50 to 100 ft in depth, and

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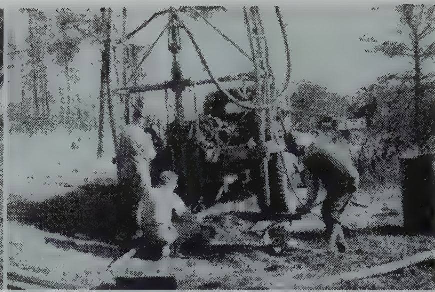

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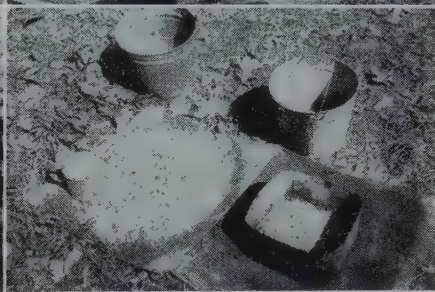

FIG 3—Surface pipe, screen, and spill box used for recovering drill cuttings.
FIG 4—Driller's helpers examine cuttings for barite, and accordingly collect or discard them.
FIG 5—Cuttings containing barite are washed to remove at least part of the clay.
FIG 6—Washed cuttings, mostly barite, from a single drill hole.

the maximum thus far drilled with full water return is 185 ft. Ordinarily, three holes can be drilled and the samples prepared in a 10 hr shift.

Preliminary drilling is commonly done at intervals of about 100 ft, but dumps, pits, and forest growth usually make it impossible to follow a regular grid. In favorable ore-bearing ground, the drill-hole interval is reduced to approximately 33 ft if most of the overburden is thick, or 50 to 70 ft if most of it is thin.

On gentle slopes, water circulation is maintained with a shallow trench from the drill hole to an excavated sump, which is within reach of a 20 ft suction hose on the mud pump. On steeper slopes, where benches for the drill are bulldozed, a steel drum connected with the hole by a wooden trough must be used instead of the sump. Slopes up to 25° have been well prospected in this way.

A 4½ in. bit is used for the upper 3 ft of each hole, in which a surface pipe or short casing is set. The surface pipe, as shown in Fig 3, consists of a 3 ft bottom section and a 1 ft top section of 4 in. pipe, connected by a coupling to which a 4 in. wide steel flange is welded. The flange is driven hard against the soil at the mouth of the drill hole to ensure that all cuttings from the hole will be discharged from the top of the pipe rather than around it.

A 16-mesh rectangular-opening screen (Fig 3) 2 ft wide and 4 ft long, with a hole cut to fit over the top of the surface pipe, is laid over the trench to the sump. The water from the drill hole, with much of the clay in suspension, drains through the screen to the sump via the trench, and the screen retains cuttings of ore, rock, and unsuspended clay.

The overflow from the surface pipe is caught and directed onto the screen by a spill box designed as shown in Fig 3. The box rests on the screen. It has sides 6 in. high cut from sheet metal, and a bottom cut from 16 in. rubber belting, with a hole cut to fit snugly over the surface pipe. The bottom is split from the surface-pipe hole to the mouth of the box, and the metal sides are in two parts that overlap opposite the mouth. These features allow enough flexure for the spill box to be removed whenever desired with the rods in the hole. While the box is in use, the overlap is clamped with a large cotter pin.

During drilling (Fig 4), the cuttings are examined constantly with the use of an ordinary kitchen strainer, and are collected in a steel drum if they contain possibly economic amounts of barite. A single sample is taken from most holes that penetrate ore, and the sample represents the full thickness of the ore. Two samples per hole are taken in rare instances, particularly if unusually good ore in minable thickness is underlain by very lean ore. Two collecting drums are kept ready for such changes in grade.

When the cuttings from an entire ore-bearing interval have been collected, the sump is cleaned out and refilled, and the cuttings are washed vigorously for about 10 min with the use of the bypass hose of the mud pump, as shown in Fig 5. The water pressure is adjusted to prevent any loss of fragmental material, and this possibility is checked by placing a strainer in the overflow from the drum. After being washed, the sample is removed from the drum and weighed wet on a beam scale. An unusually clean washed sample is shown in Fig 6, but most samples contain considerable clay. To avoid the long process of drying in order to convert the wet weight of each sample to dry weight, a standard deduction of 3 lb moisture per gallon of sample is applied to the wet weight. This average has been determined by drying tests on clean, muddy, rich, and lean samples in which the range of moisture content was found to be 2.7 to 3.3 lb per gal. After it is weighed, the sample is cut for chemical analysis.

Most of the barite in the cuttings is plus 16-mesh and minus 1 in. in size. This is far below the size range of an ordinary log-washer concentrate, showing that the barite is considerably broken by the action of the bit. An appreciable amount of the barite, however, is shattered to minus 16-mesh size, and passes through the screen. This fine ore, together with silt, is trapped in a pocket that is dug, near the sump, in the trench leading from the drill hole. This pocket is cleaned out when ore is encountered, and again after passing through the ore, and the silt collected from the ore interval is weighed after settling to eliminate excess water. To reduce analytical work, the amount of barite in the silt samples is estimated on the basis of analyses of silts from typical holes. These show a content of about 40 pct moisture and 14 to 26 pct barite. It appears safe to use an average of 18 pct (30 pct of the dry weight) and to add the amount of barite thus indicated to that shown by analysis of the screened and washed sample. A 24-mesh screen is now being tried in order to retain more of the extremely fine ore cuttings with the washed sample, and preliminary results are favorable.

Estimation of Ore Reserves

Examples of the principal data obtained from the exploratory work and related analyses of samples are condensed in Table 1. The data sheet on an actual deposit, however, might list

50 to 400 holes. Estimating the reserve of barite in a drilled tract, with the use of these data, involves the following steps:

1. From the total number of drill holes, those are selected that appear to show a body of ore that is minable under prevailing conditions. Each drilled deposit is mapped on large scale by plane table, and the surface area of the prospective mining tract is determined from the map by planimeter.

2. The total volume of bank ore is found through use of the surface area and the average vertical thickness of ore shown by the drill holes selected. The tract may be treated as a single unit if the drill-hole interval is approximately uniform, but must be broken into two or more parts if the interval varies. The volume of overburden is similarly found, and the cost of its removal is finally compared with the estimated value of the concentrate reserve.

3. The cubic yardage removed in drilling the bank-ore footage in each hole is determined, using a hole diameter 1 in. greater than the diameter of the bit. Inspection of many holes shows that this allowance may be excessive, and it may be reduced when more definite information is at hand.

4. The long-ton weight of $BaSO_4$ recovered from the bank ore penetrated in each hole is determined. This value is derived from the dry weight of the washed sample and its chemical analysis, plus an allowance for the fine ore in the silt. The proportion of Fe is determined only where its amount may be of concern in milling.

5. The optimum concentration ratio for the bank ore in each hole is found by dividing the cubic yardage of bank ore removed by the tonnage of $BaSO_4$ recovered from that volume of bank ore. This value, being based on volume rather than weight, is dependable and may be applied easily to measurements made from time to time during mining. Grade of bank ore in percentage of $BaSO_4$ by weight is less dependable, as the unit weight of the clay is not uniform and the average content of boulders cannot be determined. The curve shown in Fig 7 gives the equivalent values within narrow limits, however, and is based on an average weight of 140 lb per cu ft of barren clay in place, undried and undisturbed.

6. The average concentration ratio is obtained by weighting the concentration ratio of each hole according to the footage of bank ore penetrated in that hole, and dividing the sum of the weighted values by the total footage of bank ore penetrated in all of the holes.

7. The apparent reserve of pure barite in the tract or part thereof is obtained by dividing the total volume of bank ore found in step 2 by the average concentration ratio found in step 6. The amount may be corrected according to the anticipated grade of concentrates, but this correction is not large enough to affect the economic outlook of a deposit.

If the procedure outlined above indicates a reserve that is not in good proportion to the cost of the necessary stripping and mining, the area of the prospective tract is reduced or enlarged by eliminating or including marginal drill holes, and the apparent reserve in the adjusted area is then computed. It may be necessary to repeat this procedure several times before the most favorable mining tract is evident, and it is believed that the reserve as thus determined will not differ substantially from the actual yield.

Table 1 . . . Example of Data Used in Estimating Reserves of Barite[a]

Hole No.	Elevation	Bank Ore						BaSO4 Recovered		
		From	To	Penetration	Cu Yd Removed	OCR[b]	Ratio Weighted[c]	From Washed Sample, Lb	From Silt, Lb	Total (Long Ton)
1	824.0	13	70	57	0.2337	4.49	255.93	101.8	15.0	0.0521
2	834.5	16	68	52	0.2132	3.81	198.12	110.4	15.0	0.0560
3	827.2	16	60	44	0.1496	11.51	506.44	23.1	6.0	0.0130
4	829.0	18	56	38	0.1387	6.48	246.24	39.9	8.0	0.0214
				191			1206.73			

$$\text{Average concentration ratio} = \frac{1206.73}{191} = 6.3$$

$$\text{Apparent reserve of barite in long tons} = \frac{\text{total cu yd bank ore}}{6.3}$$

[a] Condensed from form ordinarily used.
[b] Optimum concentration ratio (cubic yards of bank ore in place per long ton of barite).
[c] Product of "OCR" and "Penetration."

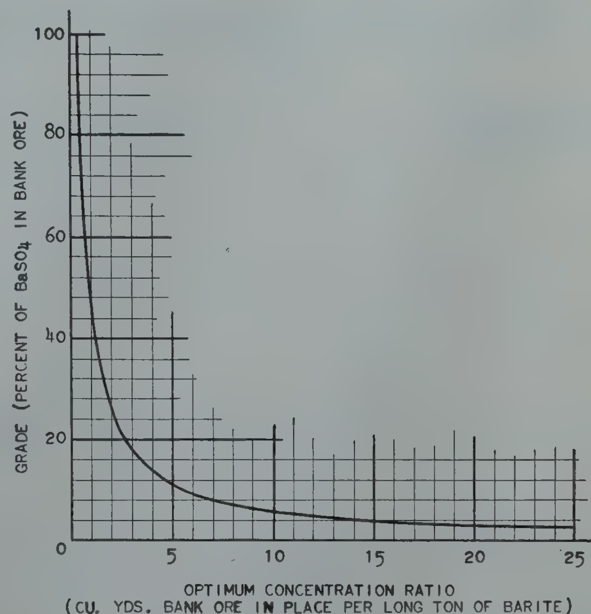

FIG 7—Curve showing relation of optimum concentration ratio to grade of barite bank ore at Cartersville.

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The Pegmatites of Jasper County, Georgia

By LENDALL P. WARRINER* and BLANDFORD C. BURGESS,* Members AIME

Location and Accessibility

Jasper County lies just north of the geographical center of Georgia, bounded on the west and north by the Ocmulgee River. The county seat, Monticello, is approximately 65 miles east-southeast of Atlanta and 40 miles north of Macon. It is served by the Macon-Athens branch of the Central of Georgia Railway Co. and by five paved highways converging from adjoining county seats. Farming and lumbering are the principal local occupations.

About half the county is covered with pine and mixed hardwoods; a large part of the forested land is owned by the Soil Conservation Service of the Department of Agriculture in its Piedmont Land Utilization Project. The topography is one of gently rolling field and forest with occasional hills rising 100 ft or more above the creek beds. Though annual rainfall is in the neighborhood of 60 in., most of it falls in torrential downpours of short duration which tend rapidly to erode the red soil and convert the country roads to greasy tracks of red mud.

History

The pegmatites of Jasper County produced a small amount of mica in World War I. In World War II, our Government's search for this strategic mineral brought to Georgia B. C. Burgess, then Southern Manager of Colonial Mica Corp., subdivision of Metals Reserve Corp., who noted on State Highway 83 the presence of pegmatitic material used as road metal.

In 1944, after severing his Government connection and returning to private practice, Mr. Burgess investigated the source of the pegmatitic material noted on his previous visit and discovered that graphic granite pegmatite had been mined for the road metal. He secured mineral leases from the Soil Conservation Service and from private owners and began preliminary development work. By 1946 he had uncovered enough pegmatite to warrant an enterprise to undertake economic development of the feldspar. In March 1947, he conveyed his leases and interest to Appalachian Minerals Co., remaining with it as Vice President and General Manager.

This company has carried on additional development, removing overburden by bulldozer and drilling the pegmatites uncovered. A modern and efficient feldspar grinding plant to produce 20 mesh feldspar for the glass industry was completed and placed in operation May 1, 1948.

St. Louis Meeting, October 1948.
TP 2620 H. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Nov. 30, 1949. Manuscript received Oct. 19, 1948.
* President, and Vice President and General Manager, respectively, Appalachian Minerals Co., Monticello, Ga.

Geology

GENERAL GEOLOGY OF ROCK TYPES OTHER THAN PEGMATITE

Jasper County lies within the Appalachian Crystalline Complex and approximately 40 miles north of the "Fall Line," where the Upper Cretace-

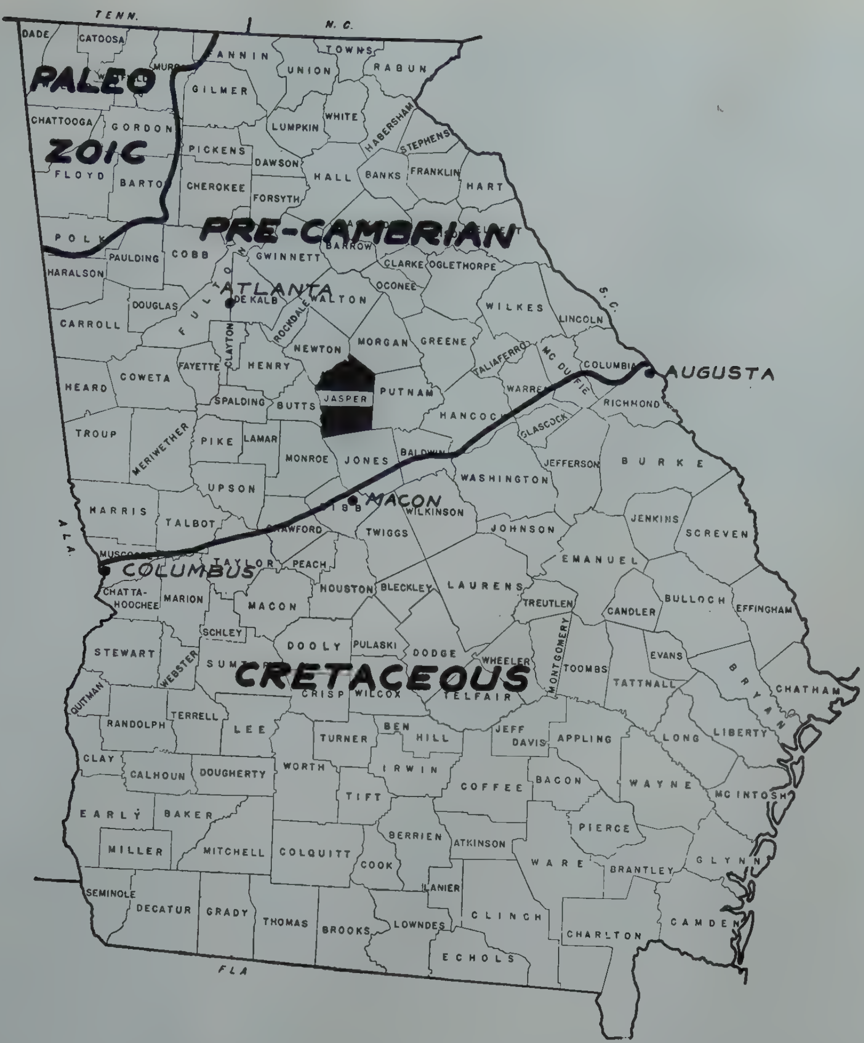

FIG 1—County map of Georgia showing location of Jasper County with respect to geologic provinces.

ous sediments of the Coastal Plain lap against the gneisses* of the Piedmont (Fig 1).

The southern half of the county is underlain by hornblende gneiss of igneous origin. Its composition approximates 50 pct hornblende in granular crystals and 50 pct plagioclase, with very minor biotite, garnet, magnetite, and chlorite, the last from decomposition of the hornblende. Originally it was probably a diorite. Within Jasper County only this rock is known to play host to the pegmatites.

The northern section of the county is underlain by augen gneiss, with a small northeasterly-striking tongue of granite gneiss cutting the augen gneiss in the northwest corner. Mica-bearing pegmatites have been located and worked in the augen gneiss in Monroe County, adjoining Jasper to the west and south.

A small granite plug intrudes the granite gneiss near the tip of the tongue, and another, used by Jasper County as a source of crushed stone for its roads, is intrusive into hornblende gneiss in the southeast corner of the county.

A number of doleritic dikes strike approximately N15°W, crosscutting all the major rock types including the pegmatites. A narrow dolerite dike cuts pegmatite at the Heard mine without causing either displacement or alteration, although fracturing in the feldspar along the boundaries appears to be oriented parallel to the strike direction of the dolerite. The dolerite itself weathers in blocky fragments which on exposure tend to spall and form round boulders locally termed "niggerheads."

The gneisses are deeply weathered and decomposed to as much as 40 ft in places with the soil zone seldom thicker than 4 ft. The granite and dolerite are much less weathered, the surface zone displaying a foot or two of soil, followed by broken fragments with oxidation extending downward along joints and fractures for only a few feet. Fig 2 shows the geology of Jasper and surrounding counties.

GEOLOGY AND MINERALOGY OF THE PEGMATITES

Mineralogy

The mineralogy of the Jasper County pegmatites is simpler than that of most deposits of this type. Rare minerals have not been noted, and minerals other than the feldspar, quartz, and mica are limited almost entirely to the iron minerals; garnet, magnetite, and ilmenite in that order of abundance.

Microcline occurs most frequently, with some albite in close association. Pink microcline and white albite in 1 in. crystals, interspersed with gray translucent quartz, show up beauti-

* Geologic map of Georgia, 1939.

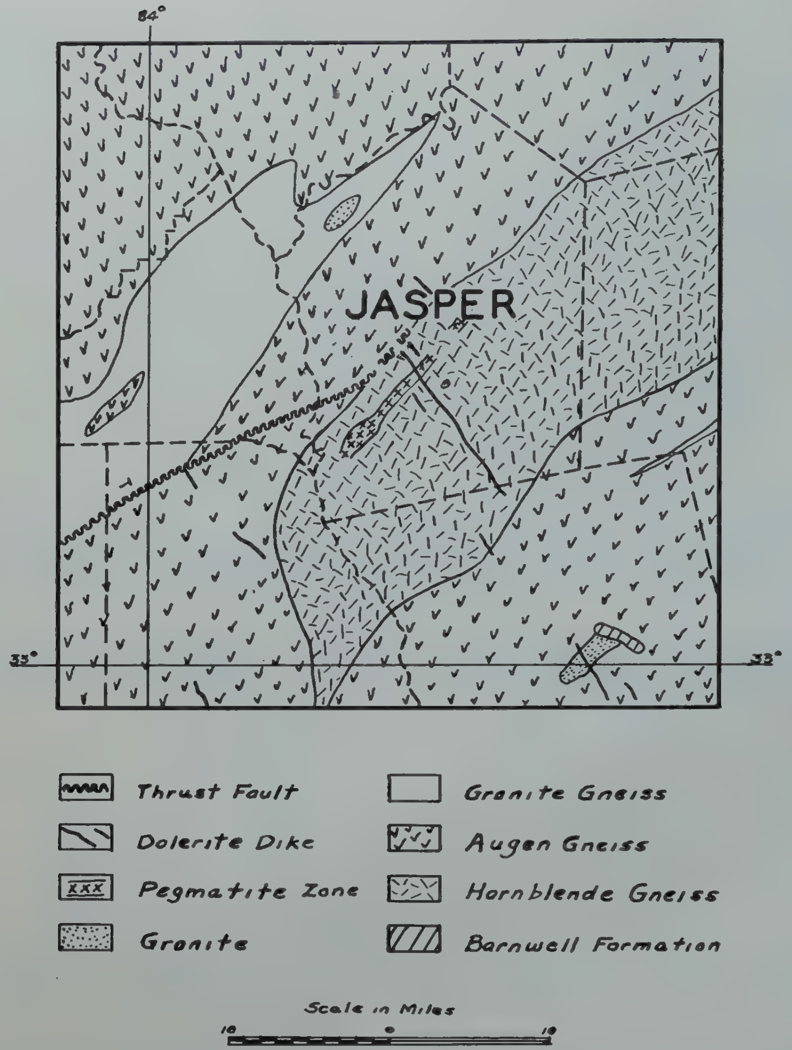

FIG 2—Geology of Jasper and surrounding counties.

fully in the freshly blasted face of the Comer No. 2 mine. In general, however, the microcline in crystals and masses up to 2 ft or more across obscures the occurrence of albite, particularly near the surface where the microcline has been bleached to a creamy white color. Below outcrop, the microcline is a delicate flesh pink, with many specimens showing a pink core fading outward to white. One occurrence of light green amazonite has been located; it too fades on exposure to sunlight.

Quartz, the second most abundant mineral, forms occasional, prominent "cores" in the pegmatites. Although none of these cores has been fully exposed yet, there are massive segregations in lenticular form up to 100 ft in length by 40 ft in width. This quartz is generally white and of a sugary texture. Elsewhere the quartz occurs in stringers and veinlets in the mass of pegmatite or along the walls. For the most part, however, it is present as grains or irregular masses in the pegmatite.

Mica appears as muscovite and vermiculite. The largest book so far located weighed 34 lb, but unfortunately displayed severe "ruling." Mining has not yet progressed far enough below the weathered zone to uncover solid block mica, but some salable pattern and punch mica is recovered from the feldspar mining operations. The muscovite generally runs in small books about 2 by 3 in. with rounded outlines. Smaller crystals display near-perfect hexagonal prisms. In color, the muscovite varies from pale greenish-white to a very dark green, but the very pale variety is limited to the vicinity of the amazonite feldspar. It forms aggregates with quartz, generally where crystallization of the pegmatite appears to have been interrupted by chilling. A feature of its occurrence with a fairly uniformly-sized ground mass of quartz and feldspar is long thin streaks radiating like a sunburst from a common center and extending outward for as much as 6 ft. This type of structure carries some vermiculite, which, in the graphic granites, occurs in identical structure. Both the muscovite and vermiculite occurring in this fashion are highly fractured and tend to break in small pieces. Partial crystals, half muscovite-half vermiculite, show a vermiculite core extending shallowly into the muscovite. Dana* states that biotite "is not infrequently associated in parallel position with muscovite, the latter, for example, forming the outer portions of plates having a nucleus of biotite." The vermiculite, then, reasonably can be attributed to the alteration of biotite. It is most abundant in the graphic granites, in plates up to 8 in. in length and 5 in. across. Outside of the pegmatites proper, it has been developed as fine flakes in the gneiss along the pegmatite borders and where small inclusions of gneiss lie wholly within the pegmatite. It may be the result of weathering.

The minor minerals, spessartite and almandite garnet, magnetite, and ilmenite, seem to be distributed heterogeneously, although the last two have been noted only in the graphic granites. The spessartite garnet has been found in dodecahedrons up to 5 in. in diameter, but so badly fractured and weathered as to be irrecoverable. Garnetiferous bands a couple of inches wide extend from the gneiss contact into the segregated pegmatite at the Heard mine for 20 ft or more, the garnet being of pinhead size. A limonitic stain penetrates the pegmatite for 1 or 2 in. around exposed garnets or magnetite, and a thin film of hematite coats the fractured surfaces of the magnetite. Magnetite is fairly abundant in the graphic granite as small grains, but poorly formed crystals up to 2 in. are to be found. Ilmenite, far less common, attains comparable size.

Texture

As in all pegmatites, the texture is highly irregular, varying from a micropegmatite at the chilled edges to coarse segregation generally near the center of the major dikes. Segregated masses or "ribs" of microcline in some cases

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* E. S. Dana: Descriptive Mineralogy. New York. John Wiley and Sons.

parallel the strike and dip of the pegmatites, and in others have dips and strikes bearing no relation to the attitude of the enclosing dike. Were the "ribs" to form the basis for a successful mining operation, a careful study of the structural control might be warranted, but no attempt is made to mine selectively at present. Fig 3 shows the microcline feldspar, quartz and graphic granite at the Benton property.

Chilled edges are general in these pegmatites, the minerals of the border zone being mixed too thoroughly to make hand sorting possible. The presence of a quartz core, however, almost invariably indicates large crystals of feldspar and of muscovite in association.

Although each pegmatite exhibits some form of zoning, no generalization can be made for the group as a whole. However, there appears to be a regional zoning in that from southwest to northeast the segregated pegmatites give place to graphic granites.

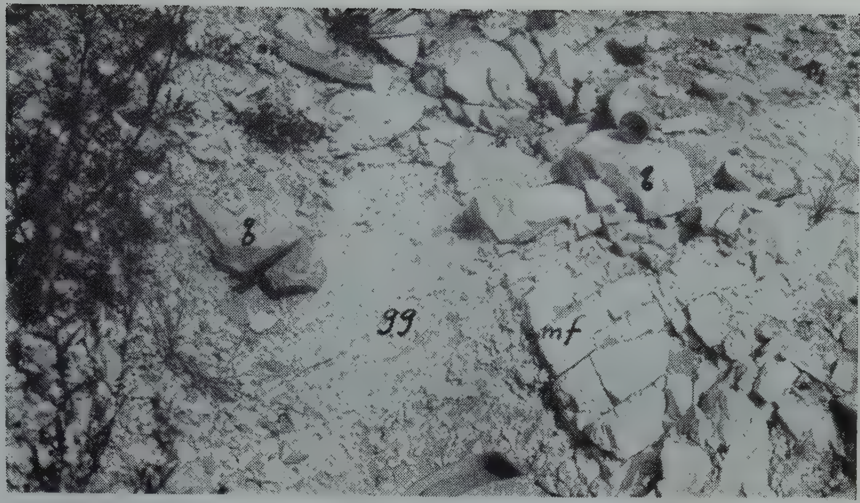

FIG 3—Microcline feldspar (mf) and quartz (q) in conjunction with graphic granite (gg), Benton property.

Origin

A more detailed regional study than we have been able to make will be necessary before valid conclusions may be drawn for the origin of these pegmatites. Their distribution indicates injection along a line of structural weakness parallel to the regional strike of the gneisses in which they occur, although local disturbance of the schistosity has taken place. Genetically and in time they are probably related to the biotite granite and granite porphyry batholith responsible for the prominent topographical feature, Stone Mountain. Continued exploration in Jasper County may uncover pegmatite in conjunction with the granite and thus afford the opportunity to study their relationship in detail.

Prospecting and Development

Residual quartz, derived from pegmatites and veins, is widespread throughout Jasper and adjoining counties. Older quartz, however, has acquired a patina of limonitic stain that distinguishes it at once from quartz weathered out comparatively recently. Fresh sugary white to gray translucent quartz generally indicates the presence of a nearby pegmatite. Heavy boulders as a rule are in place and mark the quartz "ribs" of a well-segregated pegmatite. Generally fragments of blocky microcline feldspar may be noted in the surrounding soil.

Since the pegmatites decompose less rapidly than the enclosing gneisses, they usually stand out as substantial hillocks and almost invariably outcrop on higher ground. Feldspar and graphic granite "float" mark these elevations and hillsides. It is interesting to note in this connection that nearly every pegmatite so far located was once covered by a plowed field. Although these fields have been abandoned for more than 25 years and now support a crop of 8 to 10 in. slash pine, the terraces and furrows are still visible. The former growers of cotton (the boll weevil wiped out the county's cotton in the years 1920 to 1921) may have discovered, probably by chance, that the soils derived from the weathering of high-potash microcline produced more cotton per acre.

Prospecting methods are simple. When the presence of float indicates a pegmatite, a base line as nearly as possible conforming to the long axis of the body is run out. Transverse lines are spaced at 50 ft intervals and are picketed out at right angles to the base line well beyond the estimated boundaries of the body. Soil auger holes of 4 in. diam are bored on these lines at measured intervals, with the hole locations and identity of the cuttings plotted on a base map. The soil and gravel from the pegmatites is virtually impenetrable while the decomposed gneiss is penetrated easily. The outlines of the body, and occasionally even the direction of dip, can be quite accurately determined by this method.

Removing overburden and loose outcrop material by bulldozer is the second stage if the soil auger work indicates a sizable pegmatite. As little as possible of the loose pegmatite is pushed off, as this material yields a high recovery of feldspar when washed and screened.

Detailed geological mapping follows, accompanied by shot drilling to determine the attitude and thickness of the deposit. The broken and blocky nature of the ground makes diamond drilling prohibitively expensive, with core recovery hardly better than 10 pct. Shot drilling, on the other hand, makes up for its lack of speed by eliminating time lost in fishing for lost core and in other headaches attendant upon diamond drilling in broken ground. Furthermore, because the information sought is not so much the tenor of the pegmatite as its dimensions, the high cost of diamond drilling is not justified by the information to be obtained. For example, contract prices for EX holes ranged from $2.50 to $3.50 per foot, with overburden on a cost plus basis. The company, using an antiquated Sullivan Bravo machine and with inexperienced runners drilled 13 holes ranging in depth from 22 to 69 ft at an average cost of $1.20 per foot. No. 6 and No. 7 chilled shot made satisfactory progress through material that destroyed diamond bits in as little as 10 ft.

By these methods, it is estimated that in excess of 750,000 tons of pegmatite have been proven. Probable and possible reserves are several times that amount.

FIG 4—Feldspar grinding plant of Appalachian Minerals Co. from the northeast.

Table 1 . . . Analyses of Samples of Graphic Granite and Quartz-free Samples of Feldspar

Sample No.	6	9	10	11	12
Mine	Gladesville	B. C.	Heard	Bowden	Comer
SiO_2	73.80	64.50	66.20	65.20	64.77
Al_2O_3	14.94	20.33	19.35	19.63	19.71
Fe_2O_3	0.06	0.07	0.05	0.07	0.06
CaO	0.20	0.20	0.30	0.30	1.57
MgO					0.33
K_2O	9.18	13.06	12.07	12.21	10.10
Na_2O	1.86	1.57	1.82	2.13	3.10
Loss	0.15	0.26	0.16	0.24	0.20
Total	100.19	99.99	99.95	99.78	99.84

Mining

The conventional hand method of working segregated pegmatites by sinking a shaft or open cut on feldspar-rich streaks has been discarded in favor of handling mine run pegmatite rock in a central washing and sorting plant. In this fashion, recovery of feldspar in the finer sizes is greatly increased at low cost.

Drilling and blasting practice varies with the individual deposit, but in general when a quarry face is established 8 to 10 ft holes are drilled by jackhammer on 4 to 6 ft centers and 5 to 7 ft burden. These holes are loaded with about one-half pound per foot of 30 pct extra gelatin dynamite and blasted simultaneously with No. 9 electric caps fired by a 20-hole battery. This loading tends to lift the area drilled, leaving it thoroughly broken yet with minimum fragmentation. The broken ground can then be scraped without difficulty caused by oversize lumps or solid ribs.

A long narrow cut two benches (18 to 20 ft) deep is advanced from 150 to 200 ft from the scraper set-up before side benching begins. Alternate cuts the full length of each side provide steady scraping without interfering with drilling.

Because electric power is not available at the mines, double drum Jaeger Model 2A3 hoists are used, powered by Allis Chalmers 25 hp internal combustion engines using tractor fuel. The mining method makes it unnecessary to change the position of the tail block more than once a shift, consequently the double drum hoists are adequate to do the job and represent less capital outlay with more economical operation. These hoists haul a company-designed, partial-box hoe type scraper with a 54 in. face, capacity approximately ½ ton, up a 20° steel-rail-faced ramp, depositing the broken ore on a grizzly of 56 lb rail on 10 in. centers over a 15 ton capacity wooden bin. The broken ore is trucked to the head of a central washing plant located on another pegmatite. Here production can be made from trucked material or from the pegmatite rock mined at the plant site.

At the washing plant, an Austin Western 24 in. belt conveyor raises the broken ore to a 32 in. diam by 9 ft long trommel screen with 1½ in. round openings, where it is scrubbed and washed. Trommel oversize passes to a sorting table mounted over an Austin Western 3 compartment 20 cu yd capacity steel bin. The washed feldspar is forked into the outer compartments by two men, who also remove block mica to burlap sacks mounted under funnel-shaped wooden frames on the bin deck. A feature of the sorting table is its hinged construction which permits accumulated waste to be dumped into the inner bin.

A dust jacket with ½ by 2 in. slotted openings removes fines from the minus 1½ in. from the trommel. The minus 1½ plus ½ in. is discharged from the jacket to a sorting table. Mica and quartz are removed by hand. Undersize from the dust jacket is flushed down a tail race by the wash water. The cleaned feldspar passes to the nearer of two 5-ton capacity wooden bins, the waste quartz being placed in the other.

Mixed loads of plus 1½ and plus ½ in. are trucked to the grinding plant (Fig 4) at the railroad, weighed on Fairbanks-Morse scales, and dumped into one of four 250 ton covered storage bins.

Products and Uses

Samples of graphic granite and selected, practically quartz-free samples of feldspar from some of the deposits have been analyzed with the result shown in Table 1.

The products of the sorting plant from the two mines operated by this method and from two other mines where ordinary hand sorting is practiced have given commercial feldspars of the analyses shown in Table 2.

Table 2 . . . Analyses of Commercial Feldspars

Sample No.	26	37	38	47
SiO_2	71.80	69.50	68.00	69.30
Al_2O_3	16.53	17.51	18.43	17.63
Fe_2O_3	0.07	0.09	0.07	0.07
CaO	0.20	0.20	0.10	0.10
MgO	0.04	0.04	0.04	0.04
K_2O	9.45	10.44	11.80	11.30
Na_2O	1.84	2.28	1.57	1.50
Loss	0.10	0.14	0.14	0.14
Total	100.03	100.20	100.15	100.08

The present grinding plant is limited to the preparation of feldspar for glass manufacturers.

Mining Potash Ores in Carlsbad Area

By RUSSELL G. HAWORTH,* Member AIME

Introduction

Three companies, United States Potash Company, Potash Company of America, and International Minerals and Chemical Corporation, are now operating potash mines and refineries in the Carlsbad, New Mexico, area. The three mines are located approximately twenty miles east of Carlsbad. The deposits have been found ideal for mechanized mining methods. This summary has been compiled from information made available by the three operating companies.

Prospecting

Since outcrops of potash are nonexistent, prospecting is accomplished by drilling and coring of the salt section to establish the grade, thickness, and outline of the potash beds contained in the salt deposits. Information regarding the nature of the formations in the cover of 400 to 600 ft thick over the salt and potash beds is also obtained from drill cuttings in the same operation. By proper interpretation of assembled data the best location for shafts may be determined.

The overburden varies in character but consists in general of interbedded shales, sand, gravels, limestones, and anhydrite beds. Water, in unpleasant quantities, especially in the sands and limestones, often completes the geologic section. Some of the shafts in the area have been sunk without undue difficulties, others have been delayed by flows of water. Most of the shafts are concreted from the surface to the salt section. Each of the three mines in the Carlsbad area has one shaft for hoisting ore and one for hoisting men and supplies.

Mining Methods

Information acquired from initial drilling and prospecting affected the choice of mining methods. Since the tabular, slightly rolling deposits were similar to coal seams, the room-and-pillar system used in coal mining was adopted by all the companies. Pillars have not been removed in any of the mines on a large scale. One company has conducted experimental robbing of pillars in one restricted area. Where it is known that overlying strata contain water in large amounts, it is not considered practical to risk flooding of the mines by removing pillars without fill. Some thought is now being directed toward development of economical methods of filling for pillar recovery. The problem is not identical at the three mines and local conditions will govern the methods employed for eventual pillar recovery. The important point is that, with large reserves, it has not been necessary to resort to mining of pillars for adequate production.

The ore deposits are laterally extensive but are irregular in shape and lie at a depth of 700 to 1100 ft below the surface. The major operations of all mines are on the same bed stratigraphically, except for the langbeinite level at International Minerals and Chemical Corporation. The deposits are apparently not connected and in many parts of the deposits being mined, "salt horses" or barren zones occur. From 400 to 700 ft of salt with minor beds of polyhalite and anhydrite lie immediately above the ore horizon. The ore differs from the salt only in the potassium chloride content and color due to small amounts of iron associated with the sylvite, giving the ore a somewhat mottled red and white cast. Thickness of the ore varies from 5 to 14 ft.

Rooms and break-throughs in the mines vary from 24 to 40 ft in width, the overlying salt forming an excellent roof and providing safe mining conditions. Almost no timber is used in any of the mines for roof support.

Drilling is normally the first step in mining the face of a room. Electric motor driven auger drills are used in the area. Jack hammers were used to some extent when the mines were first opened but the ore is easily drilled if bits are sharp and the coal type

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drills have been used for a number of years. Tungsten carbide tipped drill bits are being introduced now. This type of bit was first used by International Minerals and Chemical Corporation to reduce drilling costs in the harder langbeinite (hydrated potassium-magnesium sulphate) ore which is being mined in one of the upper beds.

Drilling patterns vary but are standardized at each mine. Spacing of holes ranges from 36 to 48 in. on the points of the holes. Drilling speed is 28 in. per min during actual operation.

The second step in face preparation is undercutting of the ore on or near the lower contact of the potash bearing bed. The first undercutter was introduced shortly after mining was begun in an attempt to reduce blasting costs. There are no major cleavage zones in the ore. The beds are a mass of small interlocked crystals with individual cleavages at varying angles. This produces a firm, resilient and cohesive material which is much more difficult to break than might be expected. The additional free face provided by undercutting reduces explosive costs and prevents "boot legging" of holes. Cutter bars are 9 ft in length and rounds from 8 to 8½ ft in depth are broken. The resultant smooth floor is also advantageous for mechanical loading. Cutting speed is usually 6 in. per min across the face and 5 to 6 in. kerf is cut. Shortwall undercutters are used in all three mines. Tire and track mounted universal machines are used in addition to shortwall machines at Potash Company of America.

After the face has been cut and drilled, the holes are loaded and shot with electric primers. Blasting circuits are provided for each section and blasting is done only at the end of the shift. From 0.7 to 0.9 lb of low density explosive per ton is required to break the ore.

LOADING AND HAULAGE EQUIPMENT

Advances in loading and haulage equipment have been made throughout the history of potash mining in Carlsbad. Hand loading into cars was soon replaced by the dragline or scraper loader. If hand loading were used today to handle salt and ore moved underground, over 1000 men would be required for shoveling alone. Two or three drum hoists mounted on a track-type steel loading ramp scraped the ore from the face over the ramp into cars. Track-mounted loading machines were introduced in 1936 and the trackless mining equipment was introduced by International Minerals and Chemical Corp. in 1940. These mobile, caterpillar-mounted loading machines with shuttle cars now are used by all three companies for handling most of the ore.

The loading unit is powered by one or more electric motors with a trailing cable and is mounted on a caterpillar-type truck. The loading head consists of a wide steel frame with a digging arm mounted on each side. The arms are driven by rotating plates and are pivoted on the frame. The arms dig into the pile of broken ore and sweep it on a chain conveyor in the center of the loading head. The chain conveyor operates over a long tail boom and discharges the ore into a shuttle car. The shuttle car is equipped with a chain conveyor in the bed which moves the load back from the end of the car being loaded, to the discharge end. Two or more shuttle cars are used with each loading machine. While one car is hauling ore to the discharge point, others are being loaded, thus obtaining a larger capacity from the loading machine. Discharge of ore from the shuttle cars is normally made into elevating conveyors which discharge into mine cars on a track. Ramps are often used where shuttle cars dump directly into mine cars spotted below the ramp. The shuttle cars are open on one end and the operator, by pressing the proper button, can start the chain conveyor in the bed of the car which discharges the load out the open end.

Shuttle cars range from 4½ to 11-ton capacity. The smaller cars are battery-operated while the largest types are trolley-operated. Some of the latest models are cable reel type. Batteries are designed to operate for one shift. Two sets of batteries are required for each shuttle car, one set being charged during the shift, while the other is in use.

Tracks are located at various intervals, usually parallel to the panel or working face. Maximum shuttle car haul varies from 500 to 1000 ft, depending chiefly on the capacity of the shuttle car being used.

Workings are developed so that mining progresses up-grade, if possible. Miners have known for some time that it is easier to haul down hill than uphill. Shuttle cars are sensitive in this respect also, especially the battery-powered type. Although they will negotiate relatively steep grades, it is not practical to haul ore up such grades except for short distances.

Locomotives and mine cars are of different types and sizes. Car capacity ranges from 4 to 6 tons. Locomotives from 8 to 30 ton tandem are used. Trolley locomotives are in service at all the mines. Some combination battery-trolley types are operating also. The ore is hauled to the shaft where it is dumped into the skip pockets. Rotary dumps are in service at two of the mines while the other uses Granby type cars dumping over a grizzly. The ore is crushed to minus 4 in. underground where rotary dumps are used.

VENTILATION

Ventilation systems differ in detail at the three properties but all use two shafts, one downcast, one upcast with the intake air being restricted to haulage ways from which it is coursed through working sections and return airways to the upcast shaft. One or more fans are installed either underground or on the surface along the circuit.

HOISTING

Hoisting of ore is by skip. Connected horsepower on hoists ranges up to 1000. Skip capacity ranges from approximately 5 to 8½ tons and balanced hoisting is used in each case.

All mine equipment is electrically operated and 220 volt alternating current and 250 volt direct current is used, the latter for trolley locomotives. For distribution to section transformers or motor-generator sets or rectifiers, 2300 volt alternating current is employed.

Summary

It may be noted that the operations and methods closely approximate those of coal mining due in large part to the flat beds. Nothing new has been developed in this field underground except for the adaptation of coal mining equipment to handling of the harder and heavier material. The surface plants would not appear familiar to the coal operator, however. Mining conditions in general are more favorable than most coal or hard rock mines. We are particularly fortunate in that respect. The mine superintendent's problem is no different here, though, from other mines—How to keep the bins from going empty.

A Simple Method for Making Stereoscopic Photographs and Micrographs

By LOUIS MOYD,* Member AIME

Introduction

In the preparation of illustrations to accompany reports of investigations concerning particle shapes of various natural and manufactured materials proposed for use as fine aggregates in concrete structures, it was found that stereoscopic views of such materials presented the information much more effectively than the usual two-dimensional photographs.

With the whole-hearted cooperation of the Photographic Section, a technique was developed in the petrographic laboratory of the Concrete Research Division, U.S. Corps of Engineers, at Clinton, Miss., for simple and rapid preparation of paired stereoscopic photographs and micrographs. The method requires only equipment which is usually part of the general stock of any laboratory, and it is adaptable for all purposes in which three-dimensional views would be found superior to ordinary photographs.

Stereoscopic Photographs

Where the subject or subjects to be photographed are small but not of microscopic size, an ordinary camera having a ground-glass focusing screen is set up vertically over the field. The material to be photographed is placed on a card, the surface of which is suitable as a background. The card is mounted so that it can be tilted about 5° above and below the horizontal plane, on an axis that coincides with the north-south diameter of the viewing screen. A simple way of achieving this is to mount the card on an object having a convex lower surface. In our laboratory, a watch-glass has been used, with plasticene as the mounting medium. The convex surface should be mounted on a support having a round opening on top and a flat bottom. A roll of tape, a petri dish, a beaker, or any other piece of equipment having the proper shape to give stability to the setup will be satisfactory. The arrangement of the required equipment is shown in Fig 1.

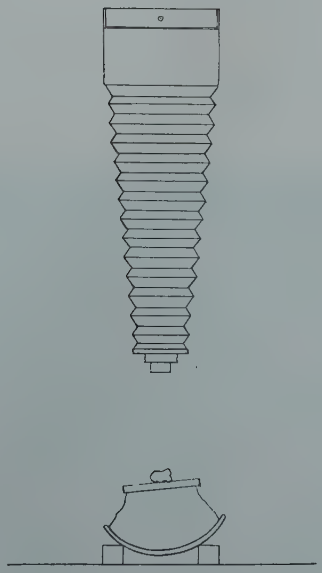

FIG 1—Arrangement of equipment for taking paired stereoscopic photographs.

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For the best results, the subject to be photographed should be mounted in the plane of the axis of rotation. To achieve this effect, the depth of the mass of plasticene between the background and the watch-glass is varied so that when the watch-glass is tilted back and forth on the axis, the subject to be photographed does not appear to move back and forth across the viewing screen. A permanent piece of equipment, embodying an adjustable vertical screw, with lock-nuts, could be made for this purpose.

The subject may be illuminated in any manner desired. It was found that the presence of shadows intensifies the effect of depth. In taking the photographs, the concave surface is mounted so that one side of the background card is about 5° below the horizontal, while the axis remains in the horizontal plane. The subject is focused carefully on the ground glass and a picture is taken. The card is then tilted so that the opposite side is about 5° below the horizontal, while the axis still remains in the horizontal plane. The subject is again put in focus and another picture is taken. After both pictures are developed and printed, they are placed side by side in the same orientation and examined with a stereoscopic viewer. If depressions appear where there ought

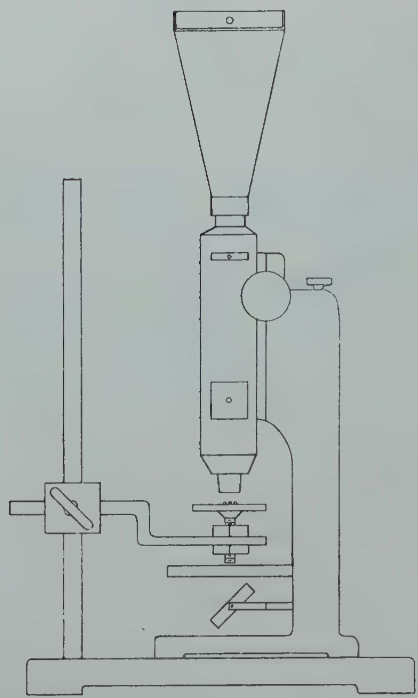

FIG 2—Arrangement of equipment for taking paired stereoscopic micrographs.

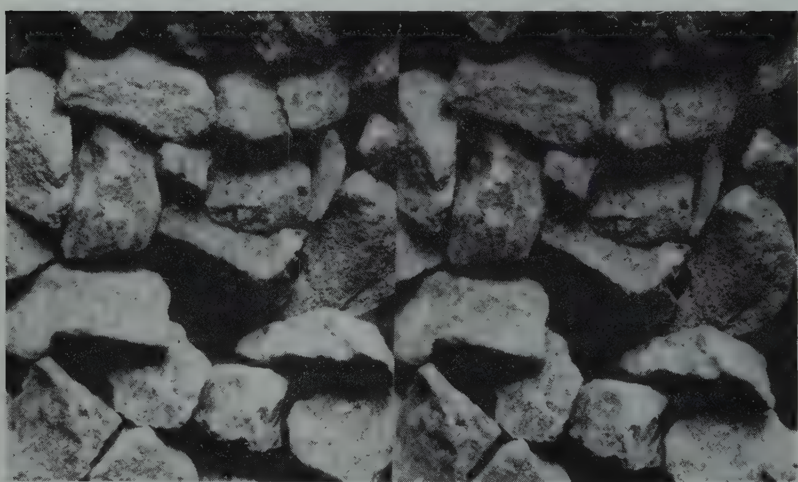

FIG 3—Stereoscopic micrographs showing shape and surface texture of sand manufactured from dolomitic limestone. Particles passed No. 8 and were retained on No. 16 sieves.

to be elevations, then the position of the pictures should be reversed. The finished pairs may be mounted together on cardboard, or the negatives may be trimmed and fastened together so that both views will appear on the same print. The paired photographs obtained by this method are equal in quality to those obtained by the use of special stereoscopic cameras or by use of any of the complex camera accessories made for the same purpose. Some of the commercially made accessories cannot be focused in the short distance required for the photography of small objects.

The procedure outlined above, involving rotation about an axis, may be modified for use with subjects too bulky for the small-scale equipment described.

Stereoscopic Micrographs

The same principle can be used for the preparation of stereoscopic photographs of subjects of microscopic size, with the method modified to meet the requirements of micrography. Any microscope and microscope-camera attachment which can be used for taking ordinary micrographs can also be used for the taking of stereoscopic pairs, and the microscope-camera arrangement is not changed for this work. The lens system should be chosen on the basis of the greatest depth of focus possible for the size of the desired field. Whereever the grain-size of the subject permitted, our laboratory used one "barrel" of a stereoscopic microscope, tilted upright, with a simple microscope-camera having a ground-glass focusing screen, thus insuring a maximum size of field and depth of focus. It is obvious from the description of the equipment used for megascopic subjects, that a universal stage on the microscope would be ideal for mounting the subject for stereoscopic micrographs. However, if a universal stage is not available, a satisfactory substitute may be readily assembled from standard laboratory equipment. Here again, it is necessary to be able to rotate the mounted subject through a range of 5° above and below the horizontal, around an axis which remains horizontal and whose direction coincides with the north-south diameter of the viewing-screen of the camera. A laboratory ring-stand with assorted clamps will satisfy these requirements. The requirement that the subject be located in the plane of the axis of rotation can be satisfied by making some simple device for regulating the elevation of the field. In our laboratory, a piece of copper tubing, bent as shown in Fig 2, was clamped into the ringstand for this purpose. The end of the tubing was flattened, a hole was drilled through it, and a small nut was soldered on. A small, flat-topped bolt was threaded through the nut and a second nut was added to lock the unit. Specimen mounts were cemented to the top of the bolt.

This simple equipment has proved adequate for the preparation of excellent, paired, stereoscopic micrographs. A typical pair of micrographs taken in this manner is shown in Fig 3. Modifications and refinements of all the above-described equipment will readily suggest themselves to anyone who might be interested in taking stereoscopic photographs by this method.

Acknowledgments

The author appreciates greatly the assistance given him by Miss Mildred Miller and Mr. Frank R. Cuddy, Photographers, Corps of Engineers.

Lightweight Aggregate Industry in Oregon

By N. S. WAGNER,* Member, and R. S. MASON,* Junior Member AIME

Introduction

The production of lightweight aggregates in Oregon is a new industry, and, like all new enterprises, it is suffering from growing pains characterized by numerous, small operations some of which flourish for a short time and then cease altogether. Normally all industrial mineral products are produced in a highly competitive atmosphere. At the present time this condition does not exist to a very marked degree in the state because as yet producers have not saturated the constantly expanding market. This paper has been prepared with the intention of outlining very briefly the current status of the various products now being used as lightweight aggregates in Oregon. The present picture will surely change, perhaps quite radically within even a short space of time.

Lightweight Aggregates Used in the Northwest

PUMICE

Interest in Oregon pumice is not new. Deposits are abundant. Successful development, however, dates only from 1946. During 1948 a total of nine operations was engaged in full or part-time production of aggregate. The postwar building boom and increased public consciousness regarding the value of insulation are the immediate reasons behind the current development. Just how firmly this production of pumice aggregate may be established is something which cannot be foretold at the present time. Much will depend upon how successfully the pumice aggregate construction already installed stands the test of time. Some fine pumice aggregate products have been made and it seems probable that because of its unique properties a certain demand for pumice aggregate will continue in the future. Fire-proof, rodent-proof, decay-resistant properties supplement the lightness in weight and insulation properties of pumice aggregate products in rendering them particularly ideal for many types of construction. In addition there has been fabrication in the form of reinforced fence posts, street markers, and similar products of a specialized-product nature and these are not to be overlooked in terms of future production. Also pertinent to how soundly pumice aggregate production is established will be possible future competition with other lightweight aggregates. However, this situation will be governed largely by competitive costs of production and marketing.

For its present use, which is almost exclusively limited to the manufacturing of building blocks, the market for Oregon pumice aggregate has been extended from the mining area around Bend and Chemult to points as far distant as San Francisco and Seattle. Naturally enough the bulk of the production goes to the Portland and eastern Oregon consumers. Shipments are made by both rail and truck.

Early in 1947 the State Department of Geology and Mineral Industries made a canvass of all pumice producers in the state. The production for 1946 amounted to 26,614 cu yd and was valued at $43,649 at the plants. The United States Bureau of Mines estimates the production for 1947 as 33,240 short tons valued at $111,380. This is roughly 65,000 cu yd. No data are available for the 1948 production but it is understood to exceed that for 1947.

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[1] References are at the end of the paper

The Oregon pumice occurrences originated largely from the eruption of Mount Mazama, the name of the formerly active volcano and the location of Crater Lake. Other lesser volcanoes throughout the area contributed to the present occurrences, however, and it has been estimated by Moore[1] that the pumice deposits cover an area of some 3500 square miles. This area lies east of Crater Lake between Bend and Klamath Falls and embraces the southern portion of Deschutes County, the northern part of Klamath County, and the northwest corner of Lake County. Thickness of the pumice ranges from thin skims to local thicknesses of as much as 30 to 40 ft. Fragment size also varies greatly. Any attempt to describe the situation by giving screen analyses would be confusing because of the wide variations to be found in an area of this size. The picture can best be summed up by stating that places showing great variations in fragment size can be found if a search for extremes is made. For mining purposes in connection with aggregate production, it can be stated that miles and miles of pumice exist in which the fragment sizes range from an inch or so downward.

The usual color of the pumice is a light gray to off-white. A typical pumice analysis shows a silica content of about 69 pct, alumina 15 pct, and sodium oxide about 5 pct. Potash, lime, and water are the next three most abundant constituents, running just a little over 2 pct each. Iron oxides are fairly constant at 2.75 pct. Titanium, manganese, magnesium, and phosphorus occur in amounts of less than 1 pct. All of the foregoing substances are combined as a glass exhibiting cellular structure. The weight of crushed but otherwise pit-run (undried) pumice runs around 1100 lb per cu yd according to figures furnished by various producers. The minimum and maximum weights per cubic yard reported are 1050 and 1400 lb, respectively. The 1400 lb per cu yd pumice contrasts with the pumice from most of the other

pits in that it is conspicuously wet when mined. No dry weight is available for this or other varieties produced.

Although lenses and interbeds of sedimentary clays and sands are present in many of the pumice accumulations, the chief impurity found in the pits is an extra-fine pink ash. This is not regarded as an impurity detrimental in nature to the product. Rather it is deliberately added to the crushed coarse pumice if the percentage of natural fines is too low. The ash therefore constitutes an impurity only when it occurs in amounts sufficient to hamper mining. Many pits show no such ash. In those that do, it occurs chiefly as overburden.

The production of pumice involves relatively simple pit and plant setups. The pumice area is traversed by both rail lines, the Southern Pacific in the Chemult area and the Great Northern at Bend, and good roads. The operations are usually either within a few miles truck haul to the railroad or immediately adjacent to it. Delivery from the standpoint of access to shipping terminals is therefore not a problem, and the situation is further eased by the fact that much of the production is shipped directly to the consumer by truck.

Mining is accomplished either by bulldozing to the plant bin, or by scoopmobiles, and in one place by a highline. As processing consists merely of crushing and screening, the plants consist of nothing more than rolls and screens. Practice varies among operations but in general the pumice is screened to about a half inch maximum. The selling of a natural or blended aggregate ranging in mesh from fines to coarse has been attended by difficulties introduced by segregation of fines during transportation. The consumer has thus been faced with the problem of drawing a somewhat classified charge of aggregate from his stockpiles rather than the desired blend. This situation has given rise to consumer complaint in the past. At least one producer who also operates a block plant is putting out a sized product which is blended to the desired proportions at the block plant. It is understood that some consumers size the mixed aggregate delivered to them to their own specifications. Drying the pumice before shipment has been considered.

No really permanent quarry and preparation installations exist at any of the pumice operations within the state. Perhaps the most striking feature about the industry is the concentration of producers in the area around Bend which is many miles to the north of the vast pumice field lying northeast of Crater Lake. The Bend area pumice deposits are scattered, limited in size, and covered with an overburden ranging from 1 to 20 ft or more. Nevertheless the proximity of the deposits to the cities of Bend and Redmond with their attendant services and supply of labor has proved to be an important factor. The Chemult area is more ideally situated than that at Bend from the standpoint of reserves, ease of mining, and proximity to two railroads. It is however in a very sparsely populated area 55 miles from Bend.

VOLCANIC CINDERS AND SCORIA

Infrequent use has been made of volcanic cinders and scoria. There are several large deposits of these materials in the state but most of them are poorly located with respect to consumption points and nearness to either rail or road transportation. Cinders and scoria when used as an aggregate in building blocks produce a heavier and stronger unit than those made of pumice.

Approximately the same specifications hold true for cinders and scoria as they do for pumice with respect to particle size. Color of the cinders ranges from red through brown to black.

HAYDITE

Haydite, an artificially expanded fossiliferous shale, is being produced in a small way at a plant about 40 miles northwest of Portland. Extensive beds of a fossiliferous, buff-colored shale occur in the area which is readily accessible to Portland and vicinity by highway and railroad. The expanded material is a brownish-red color and possesses a higher crushing strength than either pumice or scoria when used in precast concrete blocks. Haydite from this operation competes directly with pumice from central Oregon as an aggregate for lightweight building blocks in the Portland area. Higher processing costs of the haydite are offset to some extent by a smaller transportation charge (to Willamette Valley area) and by the saving of cement in the manufacture of the product since the haydite is less porous than pumice.

To be acceptable as a lightweight aggregate for concrete products, haydite must have a porous texture composed of myriads of tiny bubbles. Improper firing can produce coarse textured material having large voids, which both reduce the crushing strength and increase absorption of the cement mix.

Northwest Aggregates is currently enlarging its rotary kiln at the quarry at Sunset Tunnel on the Wolf Creek Highway, about 40 miles northwest of Portland. The plant will resume operation early in December and it is expected to produce 200 yd of finished material per day. The plant will be operated on a 24 hr day, 7 days a week, with approximately 15 men per day being employed. Pit-run material is crushed to minus 1½ in. before firing in the variable speed, 65-ft rotary oil-fired kiln. The expanded material is water cooled upon discharging and is then crushed and screened to the required sizes. The enlarged plant will be equipped with various controls to regulate feed, temperature, draft, and rotation of the kiln during the firing processes, the need for which was demonstrated in the first installation. Weight of the haydite crushed to minus ¼ in. averages about 1040 lb per cu yd, which produces a concrete block weighing approximately 40 pct less than sand and gravel units and about 1 or 2 lb lighter than pumice blocks. Crushing strength of the blocks can be varied by adjusting the amount of cement added to the mixture. Blocks with crushing strengths of from 2200 to 3000 psi are being produced. Northwest Aggregates will supply aggregate to Empire Building Materials Co., Portland, manufacturers of lightweight concrete blocks. Excess production will be available to the trade. Several monolithic concrete structures using haydite produced in the Northwest plant have been poured in the Portland area in the past few months. The material has been used as an aggregate for two poured roof slabs on large buildings with good success. Other uses for the material have included precast slabs for houses and burial vaults. Although haydite possesses a porous texture the porosity is formed by spherical, isolated bubbles rather than by tubes or voids. This condition reduces the absorption of cement greatly and also helps reduce the density of the concrete.

VOLCANIC TUFF

There are several excellent deposits of readily accessible volcanic tuff in the state but no use is being made of the material at the present time. In the past sawed blocks of tuff were used to build

durable structures in both eastern and western Oregon. Although tuff has been proven to be a durable, lightweight building stone it has not as yet been used as an aggregate for precast or monolithic concrete products.

Since volcanic tuff is composed mainly of pumice and volcanic ash, with minor amounts of clay and lumps of basaltic rocks, it offers no advantage over granular pumice, scoria, or cinders and possesses the further disadvantage of having to be crushed and sized before being used. Several short-lived attempts to produce accurately sized, sawed blocks of tuff have been made from time to time. When freshly quarried the stone can be sawed and shaped with ease, but it becomes quite hard upon exposure to the air. The main difficulties that confront a would-be producer of sawed tuff blocks are: (1) high quarrying and dressing costs plus a comparatively large capital investment for quarry preparation and stone sawing equipment, (2) relative inflexibility of operation from the standpoint of production of special sizes and shapes offered in precast units, (3) higher unit freight rates as compared to pumice, scoria, and cinders which take a lower rate because they are shipped in an unfinished state, (4) losses from damage suffered to finished blocks during handling, and (5) variation in the crushing strength of individual blocks due to differences in composition and texture. Care in selection of the rough stone would tend to eliminate this trouble but might result in the discarding of appreciable amounts of rough stone. On the other hand Oregon tuffs offer several positive advantages which place them in a better competitive position than the above disadvantages would at first seem to indicate. Tuff building stone possesses an inherent attractiveness of color, texture, and pattern that is largely lacking in precast units. A pleasing variety of contrasting stones can be readily secured by selecting material from different locations in the quarries. Cement apparently will be in short supply for some time to come, a factor favoring sawed blocks over precast units. Blocks of natural tuff should find a market in the construction of durable public buildings and private homes.

PERLITE

Perlite production in Oregon is presently confined to the operation of Dantore Products Division of Dant and Russell, Inc., on the Deschutes River in southern Wasco County. Details of this operation have been adequately described in a paper read at the El Paso regional meeting of the AIME in October 1948 by Mr. Fred Gustafson,[2] resident mining engineer of the Dantore Division. An earlier report on the property was made by J. E. Allen.[3] Recently the company announced that it will move its processing plant from St. Helens to the mine and that an insulating, noncombustible accoustical tile plant would also be erected adjacent to the beneficiation and expansion plants. Total cost of this construction has been stated to be one million dollars with completion date set for the summer of 1949.

DIATOMITE

Diatomite has been used for many years in the production of lightweight concretes and also for conditioning concrete mixtures for good insulation properties. No block manufacturers are now using diatomite in the state.

Tonnage figures of diatomite used as a lightweight aggregate in Oregon are not available. Practically all production is from one quarry in Deschutes County. The material is produced at the quarry and is sold for a variety of uses. Approximately one-eighth of all diatomite produced in the United States is used as a lightweight aggregate.

Summary and Conclusions

The lightweight aggregate industry in the state is dominated by the various pumice producers in the Bend-Chemult area of central Oregon. These producers are essentially running a materials-handling business rather than a true mining and milling enterprise. Equipment is usually simple and the crushing and screening is accomplished with a minimum of capital outlay.

Operators in the area near Bend are faced with the necessity of finding new pits from time to time since individual deposits have tended to be small or covered with overburden which varies greatly in thickness. In the Chemult area operators have an unlimited supply of material to choose from which has little or no overburden but which at the same time is unprotected from surface moisture and, during the winter months, from freezing conditions, both of which tend to place the operators in a less advantageous position than those in the Bend area.

The production of haydite is just being started but as it finds wide acceptance there would appear to be no obstacle to its continued production since unlimited deposits of suitable material are readily accessible.

The production of tuff in the form of sawed blocks could offer direct competition to other lightweight aggregates provided several production difficulties could be ironed out. It is doubtful however if tuff blocks will ever find the same widespread acceptance that other lightweight aggregates are currently enjoying.

The use of expanded perlite and obsidian appears to have great promise. Whether perlite will be used widely as an aggregate in precast concrete blocks remains to be seen but its continued and expanding use as a lightweight plaster sand and in lightweight insulating wallboards seems practically assured.

The use of diatomite in lightweight concrete is not new but will probably always be limited to a few specific applications.

For the first time in the history of the Pacific Northwest the construction of wood frame buildings is slowly yielding to the inroads of lightweight concrete structures utilizing either precast units or monolithic construction. This change has been very gradual in the past but now appears to be accelerating. The change has been brought about by a number of factors, chief among them being the current scarcity of wood for construction coupled with high heating costs for frame buildings and the desire on the part of many builders to erect more durable structures.

At present the market for all of the lightweight materials appears to be unlimited but this condition will undoubtedly change in time. Eventually the picture will be one of a few well-established, carefully run operations for each of the various products which have tailored their production to a market that has been fairly well defined but which will be increasing steadily for a number of years.

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Economics of Coal for West Coast Power Generation

By CLAUDE P. HEINER,* Member AIME

While the title of this paper embraces the entire West Coast, the author, in the interest of simplification, has confined the discussion to California—particularly the central section.

California's population has risen 45 pct since 1940. Its electric requirements have also increased, not only because of the growth in industry and population but also because of the widespread tendency of all types of electric consumers to use more electricity than ever before. The growth of California's electric utilities is shown by published data covering operations of three of its large systems and is given in Table 1.

New Capacity Steam Driven

In meeting these additional electric loads there has been a substantial increase in generation from fuel burning plants as shown by the published energy statistics given in Table 2.

Increased Fuel Oil Consumption

A study of construction plans of the three electric utility systems named in Table 2 clearly shows that expected new electric loads are to be met, to a large extent, by new steam generating capacity as shown by Table 3. D. D. Smalley, Vice President of the Pacific Gas and Electric Co., in an address before the Pacific Coast Electrical Association in June 1948, stated that California utilities plan to meet new electric loads through greater use of steam plants and that in 1951 approximately 45 pct of the area's electric production would be by steam as compared to 33.6 pct in 1946. He stated that if load growth develops as expected and if the year 1952 is dry, thereby requiring high load factor steam operation, the Pacific Gas and Electric Co. would need the equivalent of nearly 25 million barrels of fuel oil.

Federal Power Commission reports show that the energy input to the Pacific Gas and Electric Co. system was approximately one billion kilowatt-hours greater in 1947 than in 1946. They also show that the steam plant operating capacity factor in 1946 was approximately 35 pct; in 1947 it was 72 pct due, probably, to the extreme drought.

Table 3 shows that Pacific Gas and Electric Co. now has under construction 327,000 kw of hydro capacity which, by 1950, will bring its total hydro generating capacity to 1,313,899 kw. In addition to hydro capacity under construction that company also has 975,000 kw of steam generating capacity under construction, which by 1951 will bring its total steam capacity to 1,560,834 kw. If it is assumed that this total steam electric capacity is operated at a capacity factor of 45 pct in 1951, over 6 billion kilowatt-hours would be generated. If the load growth on the Pacific Gas and Electric system continues at the current rate of approximately one billion kilowatt-hours per year and if it is assumed that 70 pct of this increase would be generated by steam (30 pct assumed to be generated by hydro facilities with average water conditions), the steam electric production in 1960 would amount to 12 billion kilowatt-hours, which would require fuel the equivalent of 24 million barrels of oil or 6 million tons of coal.

San Francisco Meeting, February 1949.
TP 2692 F. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Dec. 30, 1949. Manuscript received Jan. 28, 1949.
* President, Utah Fuel Co., Salt Lake City, Utah.

Government Projects Inadequate

Much has been written with respect to the plans of the Department of Interior in regard to the installation of hydroelectric plants as a part of multipurpose dams relating to reclamation, flood control, and power generation. An examination of the effect of such plans on the power supply for central and northern California is of interest. The Department of Interior now has five 75,000 kw units at its Shasta hydro plant on the Sacramento River. It now appears that its three 25,000 kw unit Keswick hydro plant on the same river will be available during the winter of 1949 to 1950. Its estimates show that in a dry year there is sufficient water for operation of Shasta units at an average load factor of 36 pct and Keswick units at a load factor of 41 pct. It is also giving consideration to the development of the Pine Flat and Haas hydroelectric plants on Kings River for generating capacities of 45,000 kw each. The total installation of Government units in Central Valley and Kings River in California is relatively small when compared with the

fast-growing electrical requirements of the area.

Obviously the Department of Interior recognizes the effect of drought on the output of these plants inasmuch as its plans include installation of a 240,000 kw steam plant at Delta east of Berkeley and a 90,000 kw steam plant at Fresno.

More Colorado Power Remote

To supplement California's electric power supply, the Department of Interior plans ultimate development of projects of considerable magnitude on the Green and Colorado rivers but these plants would be extremely expensive and, inasmuch as they are very remote from the West Coast, the cost of transmission facilities would be great. Furthermore final division of Colorado river waters among the respective upper and lower basin states would be necessary, and because construction progress would depend on year to year appropriations by Congress, if the projects are approved, a number of years will be required, in the author's opinion, before electric energy will flow from these sources to California.

California Faces Oil-gas Importation

For the past 25 years California has not only supplied all of its fuel oil requirements but those of other West Coast States and Arizona, Nevada, and to some extent Idaho, as well as a large portion of the requirements of China and Japan. California is now confronted with the importation of oil.

It has for many years supplied its populace with large quantities of natural gas. However, during the past year the El Paso Natural Gas Co. constructed a line from the west Texas field to Blythe, Calif., for the ultimate delivery of 305 million cubic feet per day to supplement southern California supply. To supplement the locally available dry natural gas supply from northern California fields the major natural gas company serving that area has under consideration the purchase of 100 million cubic feet of natural gas per day from southern California gas companies to be delivered through the El Paso line for the period ending 1953. In addition to this temporary arrangement the northern California gas

Table 1 . . . Growth of Electric Utilities in California

Name of Utility	Electric Energy Sales,[a] Billion Kilowatt-hours		
	1940	1947	Increase, Pct
Pacific Gas and Electric Co.	4.7	8.5	81 (7 yr)
Southern California Edison Co.	2.9	5.6	93 (7 yr)
	Fiscal Year Ending June 30		
	1940	1946	Increase, Pct
City of Los Angeles	1.49	2.56	72 (6 yr)

[a] Delivered to customers' meters.

Table 2 . . . Increase in Generation of Electric Energy

Name of Utility	Electric Energy Generated Billion Kilowatt-hours		
	1940	1947	Increase, Pct
Pacific Gas and Electric Co.			
From hydro plants	4.24	4.90	16
From steam plants	0.45	3.71	822
Purchases from and net interchanges with other utilities	1.37	1.81	32
Southern California Edison Co.			
From owned and leased hydro plants	3.45	4.03	17
From owned steam plants	0.14	2.40	1,715
Purchases from and net interchanges with other utilities	0.05	0.23	460
	Fiscal Year Ending June 30		
	1940	1946	Increase, Pct
City of Los Angeles			
Los Angeles units at Hoover Dam	1.73	2.85	65
From owned hydro plants	0.25	0.42	68
From owned and leased steam plants	0.06	0.20	333

Table 3 . . . Comparison of Present Electric Generating Capacity with that now under Construction

System	Electric Generating Capacity, Kilowatts			
	In Service at End of 1947		Installed in 1948 or Under Construction at End of 1948	
	Hydro	Steam	Hydro	Steam
Pacific Gas and Electric Co.	986,899	585,834	327,000	975,000
Southern California Edison Co.	904,520[a]	396,000	35,000	255,000
City of Los Angeles	621,025[b]	403,800[c]		225,000[d]

[a] Including 495,000 kw leased at Hoover Dam.
[b] Including 495,000 kw leased at Hoover Dam.
[c] Including 86,300 kw leased on "when as and if" basis from Southern California Edison Co.
[d] Reference 45th Annual Report of Board of Water and Power Commission, City of Los Angeles, for fiscal year ending June 30, 1946.

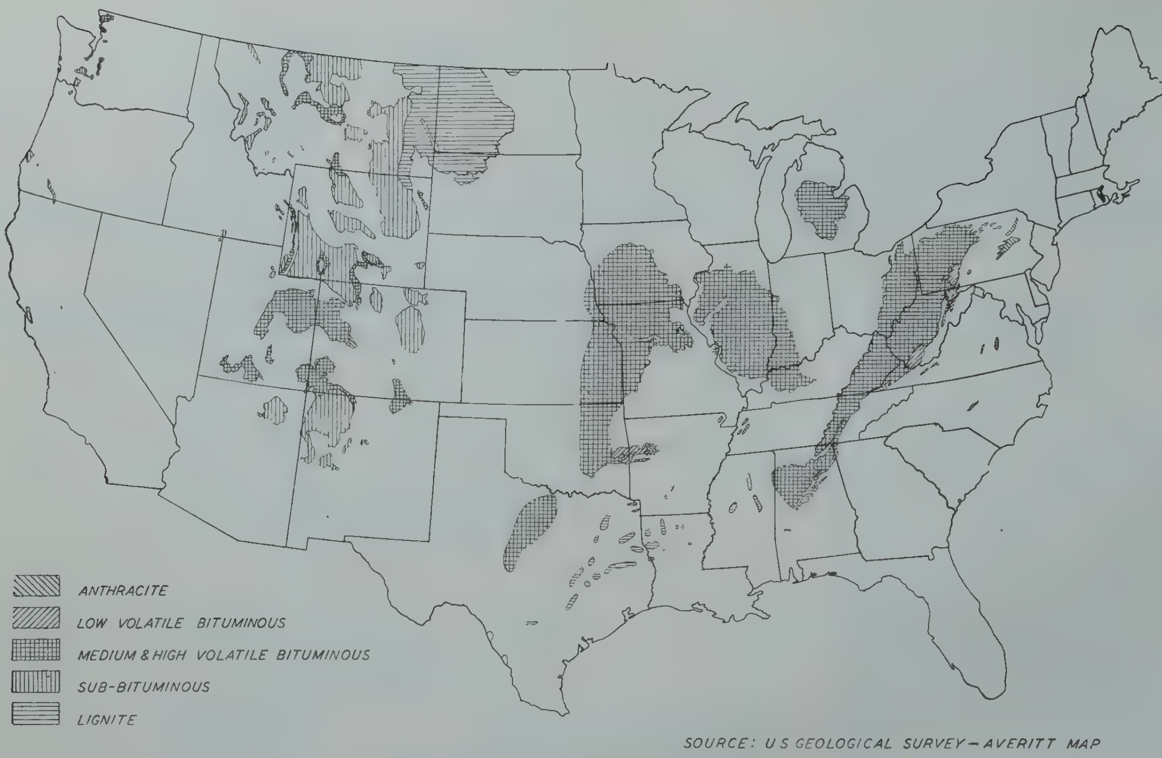


FIG 1—Coal fields of the United States.

service company has entered into a preliminary agreement with the El Paso Natural Gas Co. for the purchase of 100 million cubic feet of natural gas per day for a term of 25 years, to be delivered at the California border near Needles and, at its option, to increase purchases up to 300 million cubic feet per day by 1954. It is understood that the El Paso company plans the construction of a new line from San Juan, N. M., to Needles, Calif.

It is generally believed that during the greater part of the year natural gas transported into California will be used to meet the domestic and commercial requirements of the state's fast-growing population and that little gas will be available for large industries. Because of the desire of natural gas distributors to reserve such fuel for domestic and commercial use it is believed that use of natural gas for electric power generation, even on a dump basis, will likely decline.

Fuel Oil Immediate Basic Fuel

For the immediate future fuel oil likely will be used as a principal fuel for California steam electric generating plants and, because of the general increase in consumption of fuel oil in the United States, this oil may be supplied indirectly from foreign sources such as South America and the Middle East. Consideration has been given to the exportation of west Texas crude to California via pipeline which would require that eastern seaboard markets, now supplied from Texas, be supplied from some other source, probably South America, at such time as Arabian crude can meet the European crude market currently being supplied by South America. On the other hand much has been written during the past 30 years concerning the rapidly declining fuel oil reserves of the United States. The available known reserves of crude oil at existing rates of consumption have for many years been variously estimated at from 15 to 30 years but it is understood that at this time proven oil reserves of the United States, at present rate of consumption, are greater than at any time during the past 25 years. However, it is likely that during the years to come domestic crude oil will come from deeper horizons which will necessitate increased development costs and higher recovery costs.

Activities of the major oil companies and the Federal Government indicate cognizance of the fact that it will be necessary to find new sources for oil and gasoline within some foreseeable future time, as indicated by the following plans:

1. Supplementing reserves with imports from recently discovered oil fields of South America and the Middle East.
2. Use of synthetic conversion of gaseous fuel to liquid fuel.
3. Research and construction of pilot plants to convert coal to gas and to synthesize this gas resulting in the manufacture of gasoline and fuel oil.
4. Extraction of oil from western shales.
5. Exploration for possible increased oil production from the continental shelves.

Fuel Oil Prices May Increase

Imports of fuel oil from foreign sources will likely increase the price of fuel oil. Greater consumption of gasoline and diesel fuel and improvements in refining processes, resulting in much higher recovery of these commodities, will also tend to increase the price of fuel oil.

While it is believed that the plan of supplementing domestic fuel oil reserves with imports from South America and the Middle East may be economically feasible in a peacetime economy, it is very doubtful if it would be practical in case of national emergency. When consideration is given to the great demand for petroleum pro-

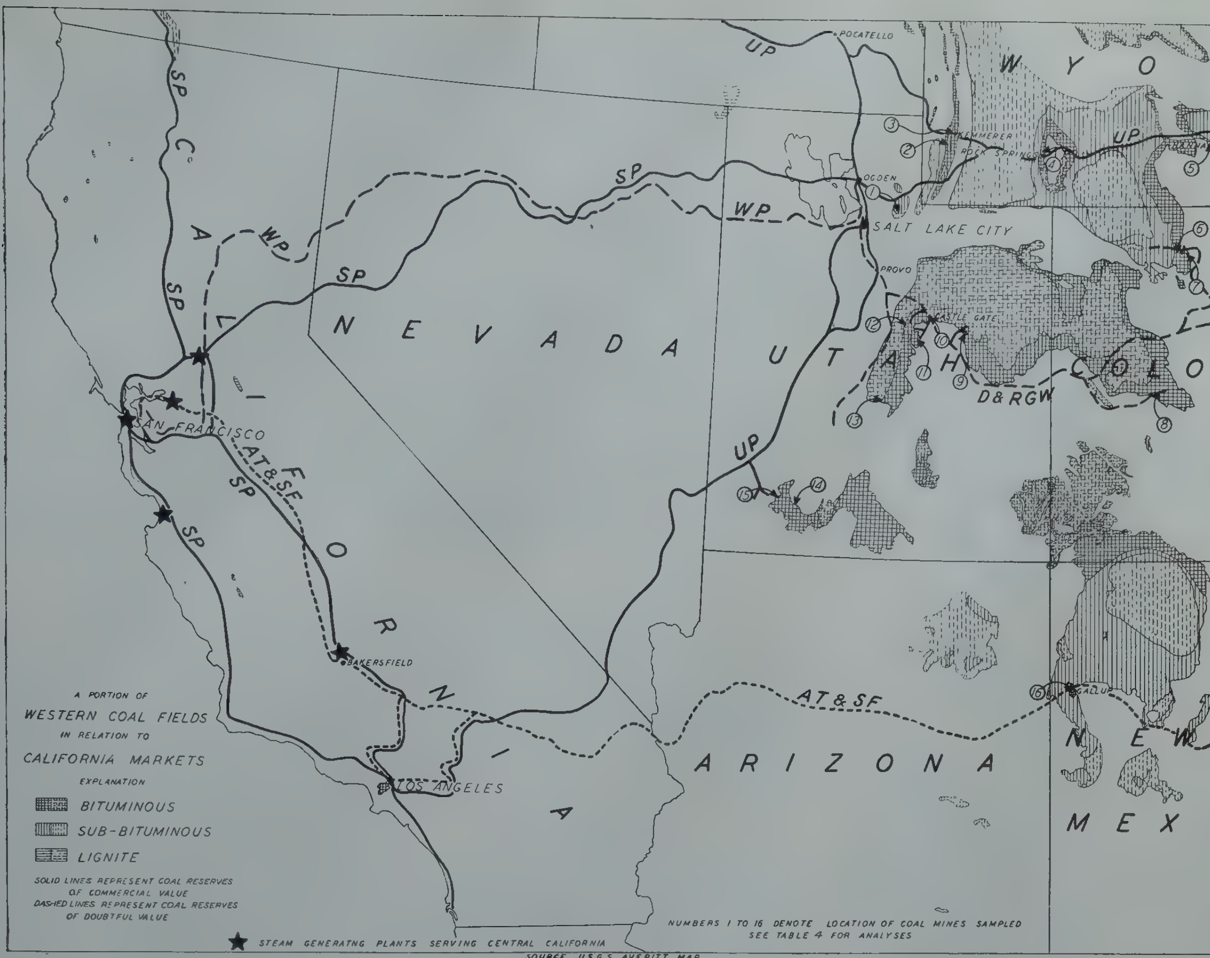


FIG 2—A portion of western coal fields in relation to the California market.

ducts during the past war and the burden of such demands on the United States together with the present day accelerated use of oil for general purposes, it is believed that the availability of oil for the generation of electric power in California would be somewhat precarious in event of war.

In discussing the use of oil and natural gas for generation of electric power in California it is interesting to note the opinions of electric utility managements operating in north central Texas in connection with the design of steam electric generating plants. The vast reserves of crude oil and natural gas in Texas are well known and for years natural gas has been the principal fuel for the generation of electric power in that state with oil as an adjunct for use during extremely cold weather or emergencies. Plants of major utilities in north central Texas are now being designed for ultimate conversion to coal, because it is the opinion of managements of those utilities that within the useful life of the plants, natural gas for the generation of electric power will become unavailable due to exhaustion or be made unavailable for industrial purposes by state legislative action. In the event of such legislative action it is believed that the state would likely curtail out-of-state shipment of gas prior to imposing restrictions as to use within its own boundaries.

It is understood that California utilities are now constructing steam plants for possible ultimate conversion to solid fuel.

Ample Coal Reserves Available

Fortunately, with the decline in oil reserves within the foreseeable future, the United States has vast solid fuel reserves. The accompanying map, Fig 1, shows the location and the general quality of available coal reserves within the United States. It will be noted that a large portion of these reserves is in the western states and Fig 2 is an enlargement of this latter area to show the relationship of coal reserves of southwestern Wyoming, western Colorado, Utah, and northwestern New Mexico to possible California markets. This map shows the location of coal of known and doubtful commercial value. It also shows the quality of the coal. Coals from this region are of excellent quality as is indicated by analyses of typical samples from the more important fields given in Table 4. Moisture, ash, and sulphur contents of the bituminous coals are low and the heat value on an "as received" basis ranges from 11,000 to almost 13,000 Btu. Ash fusion temperatures are higher than those of some midwestern coals.

Because of the distance from the intermountain states to central California and increases in the price of coal since the war it is probably the general opinion that, if coal were used instead of oil for the generation of electric energy, there would be a considerable increase in production cost. Actually the difference would not be great. While there has been a gradual increase in the mine price of coal since the war, there has been a much greater increase in the price of fuel oil.

Table 4 . . . Typical Analyses of Coal from Mines in Parts of Western States

Map Reference Number[a]	State	County	Mine	Size	Analysis by[b]	Analyses on "As Received" Basis, Pct					Btu	Ash Softening Temperature, °F	Grindability (Hardgrove)
						Moisture	Volatile	Fixed Carbon	Ash	Sulphur[c]			
1	Utah	Summit	Coalville	1 × ¼ in. slack	A	14.4	36.0	43.9	5.7	1.3	10,690	2,140	38[d]
2	Wyoming	Lincoln	Elkol Bed	1⅝ in. × 0 slack	A	19.9	35.1	42.2	2.8		10,430	2,260	
3	Wyoming	Lincoln	(Kemmerer Dist.)	Slack	A	5.5	35.7	46.5	12.3	1.0	11,980		45[d]
4	Wyoming	Sweetwater	(Rock Springs Dist.)	1⅝ in. × 0 slack		11.5	35.7	48.4	4.4	1.0	11,700	2,090	35[d]
5	Wyoming	Carbon	Elk Mountain	Slack	A	7.8	41.5	41.5	9.2	0.6	11,380		40[d]
6	Colorado	Routt	Moffat	1½ in. × 0 slack	A	8.5	38.5	48.5	4.5	1.1	12,230		35[d]
7	Colorado	Routt	Edna Strip	1¼ × ¼ in. pea	B	8.1	40.7	47.6	3.6	2.0	12,232	2,025	35[d]
8	Colorado	Gunnison	Somerset	1½ in. × 0 slack	B	5.4	37.5	51.1	6.0	0.6	12,885	2,510	51.8
9	Utah	Carbon	Sunnyside No. 1	1 in. × 0 slack	B	6.3	36.8	50.2	6.7	1.2	12,932	2,750	53.9
10	Utah	Carbon	Castle Gate	1 in. × 0 slack	B	6.3	40.9	46.3	6.5	0.5	12,586	2,169	47.0
11	Utah	Emery	King	1 in. × 0 slack	A	6.2	42.4	44.4	7.0	0.6	12,590	2,310	48[d]
12	Utah	Carbon	Clear Creek	1 in. × 0 slack	B	8.0	39.9	46.7	5.4	0.5	12,340	2,520	42.8
13	Utah	Sevier	Salina Canyon	1½ in. × 0 slack	B	7.6	41.2	43.3	7.9	0.4	11,530	2,095	42[d]
14	Utah	Garfield	Alvey	2 in. lump	A	17.7	36.2	38.3	7.8	0.4	9,840	2,280	38[d]
15	Utah	Iron	Webster	1⅝ in. × 0 slack	A	6.1	39.7	39.6	14.6	6.1	10,670	2,150	42[d]
16	New Mexico	McKinley	Mutual	"Tipple"	A	13.8	34.1	40.2	11.9	0.5	10,360		35[d]

[a] Refer to Fig 2.
[b] *A*—U.S. Bureau of Mines
B—Utah Fuel Co. Laboratory
[c] Separately determined.
[d] Estimated by Author.
[e] Estimated as being general average of slack coal as shipped from Rock Springs; original data from sources believed to be reliable.

Lower Freight Rates Possible

As developed later in this paper, there is considerable difference in freight rates for the shipment of coal for domestic use and for export from the intermountain region to West Coast points. The freight rate on coal to be exported is lower and it is the author's opinion that the export rate could be obtained to deliver coal into central California for the generation of electric power. The curves in Fig 3 show the trend since 1940 in prices of coal and oil delivered to the San Francisco Bay area. Prices are in cents per million heat units (Btu) in the fuels based on published prices at the end of each year assuming the figures given in Table 5. It will be noted that coal was considerably higher than oil until 1947 at which time there was little difference.

Table 5 . . . Cost of Shipping Coal and Oil

Coal	Oil
(*a*) 25 million Btu per ton	(*a*) 6.3 million Btu per bbl
(*b*) Published freight rate plus 1 cent from Carbon County, Utah, to San Francisco on coal for export	(*b*) Delivered at Port Costa plus barging charge of $0.08 per bbl

Coal Entails More Equipment

Where coal is used to generate electric power more expensive equipment is necessary than with oil; also operating and maintenance expenses are somewhat higher. The dotted line curves on Fig 3 show what the effect of fixed charges and operating and maintenance expenses incident to burning coal and oil respectively would have been at the end of years 1946, 1947, and 1948 assuming plants would have been operated at a 50 pct annual load factor.

Boilers for steam plants now being constructed in central California are of a size that would require that coal be burned in pulverized form. Coal requires track hoppers and car dumpers for unloading; facilities to handle it in and out of storage and from railroad track hoppers to plant bunkers; bunkers for its storage above pulverizing mills; pulverizing mills, and fans to transport it to the boilers; and ash handling equipment. Precipitators to

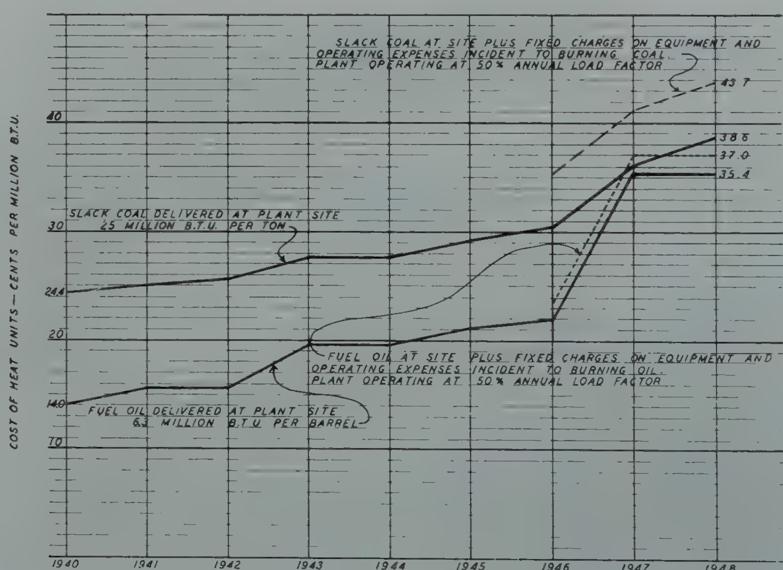


FIG 3—Trends in prices of fuel oil and coal in the San Francisco area.

Table 6 . . . *Economic Comparison of Burning Coal and Oil for Generation of Electric Power in San Francisco Area Assuming Coal Delivered at Generating Plant for $9.65 per Ton*

Cost of Operation Using Coal, per Ton of Coal Consumed with 300 M-kw Plant at 50 Pct Annual Load Factor								Cost of Operation Using Oil, per Bbl Consumed with 300 M-kw Plant at 50 Pct Annual Load Factor			
			Operating Expenses Incident to Burning Coal								
Heat Value of Coal Btu per Lb "as Received"	Cost of Coal Delivered to Plant Site	Fixed Charges on Plant Investment Incident to Burning Coal	Coal Handling and Ash Disposal	Operating Labor	Maintenance	Total Fixed Charges and Operating Expenses Incident to Burning Coal, Other than Cost of Coal at Site	Total All Costs Incident to Burning Coal	Fixed Charges on Plant Investment Incident to Burning Oil	Operating and Maintenance Expenses Incident to Burning Oil	Cost of Oil Delivered to Plant Site which with Fixed Charges, Operating and Maintenance Expenses is Equivalent to Price of Coal in Column (2)	Total All Costs Incident to Burning Oil
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)
12,500	$9.65	$0.95[a]	$0.20	$0.054	$0.06	$1.264	$10.914	$0.074[b]	$0.029	$2.622	$2.725

[a] Assumes cost of facilities incident to burning coal at $17 per kw; boiler efficiency (operating) at 87 pct; station auxiliary power consumption at 8 pct of gross generated.
[b] Assumes cost of facilities incident to burning oil at $4.50 per kw; boiler efficiency (operating) at 86 pct; station auxiliary power consumption at 7 pct of gross generated.

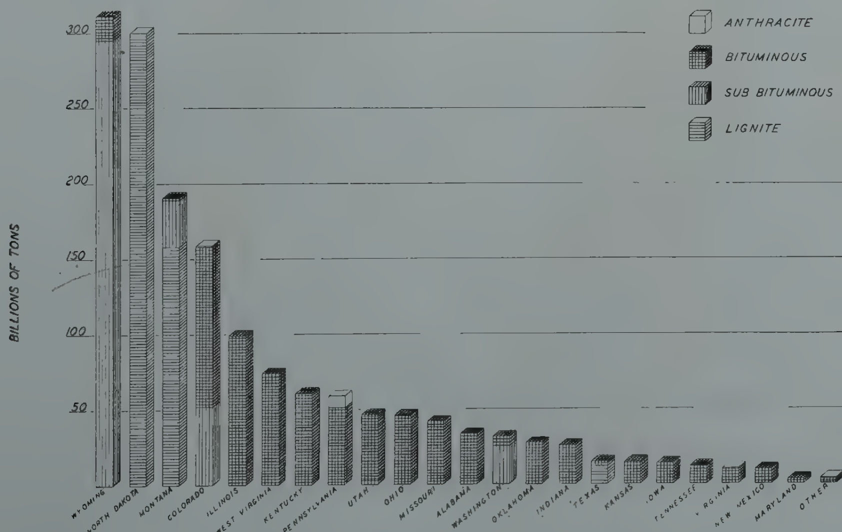
remove the fly ash from the flue gases as they leave the boilers are necessary in metropolitan areas. The cost of these facilities for plants of 300,000 kw size amounts to about $17 per kw. The cost of facilities for burning oil such as unloading pumps, storage tanks, transfer pumps, piping and preheating facilities for a 300,000 kw plant would be about $4.50 per kw.

Coal also requires labor to handle it amounting to about 15 cents per ton; labor to handle and dispose of ashes amounting to about 5 cents per ton; labor to operate fuel grinding equipment amounting to about 5 cents per ton of coal. Coal results in slightly higher boiler maintenance than oil because of ash. It is estimated that the maintenance of fuel grinding and fuel and ash handling equipment together with incremental boiler maintenance over that required for burning oil would amount to about 6 cents per ton of coal burned.

Coal Use Feasible

Table 6 shows a comparison of costs incident to burning coal and oil in a 300,000 kw plant and it will be noted that segregation is made between fixed charges, coal and ash handling expense, operating labor and maintenance expenses. The load factor at which a plant is operated naturally has a bearing on the amount of fuel consumed and the table assumes an annual load factor of 50 pct with either fuel; coal quality also has a bearing on quantity consumed and thus influences fixed charges per ton consumed; the table assumes 25 million Btu per ton. With a given boiler the efficiency with oil is about 1 pct lower than with coal because of the difference in hydrogen contents. On the other hand oil requires about 1 pct less auxiliary power because of absence of coal grinding and conveyance, so the heat consumption per net kilowatt-hour would be approximately the same with either fuel. Table 6 shows that coal delivered to a central California electric generating plant at $9.65 per ton would be equivalent in price to oil delivered at the same site at $2.62 per barrel considering fixed charges and operating and maintenance expenses incident to use of the two fuels and assuming operation at 50 pct annual load factor.

The quality of coal has an effect on



SOURCE US BUREAU OF MINES AS REVISED BY CARL E MILLER, TECHNICAL ADVISER BATTELLE MEMORIAL INSTITUTE

FIG 4—Original coal resources in the United States.

Table 7 . . . Economic Comparison of Burning Coal and Oil for Generation of Electric Power in San Francisco Area Assuming Oil at Present Price

Cost of Operation Using Coal, per Ton of Coal Consumed with a 300 M-kw Steam Plant at 50 Pct Annual Load Factor								Cost of Operation Using Oil, per Bbl of Oil Consumed with a 300 M-kw Steam Plant at 50 Pct Annual Load Factor			
			Operating Expenses Incident to Burning Coal								
Heat Value of Coal Btu per Lb "As Received"	Cost of Coal Delivered to Plant Site to be Equivalent to Oil at Present Price Stated in Column (11)	Fixed Charges on Plant Investment Incident to Burning Coal	Coal Handling and Ash Disposal	Operating Labor	Maintenance	Total Fixed Charges and Operating Expenses Incident to Burning Coal, Other than Cost of Coal at Site	Total All Costs Incident to Burning Coal	Fixed Charges on Plant Investment Incident to Burning Oil	Operating and Maintenance Expenses Incident to Burning Oil	Present Cost of Oil Delivered to Plant Site[a]	Total All Costs Incident to Burning Oil
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)	(11)	(12)
11,000	$6.794	$0.840	$0.20	$0.046	$0.06	$1.146	$7.94	$0.074	$0.029	$2.23	$2.333
11,500	7.236	0.895	0.20	0.049	0.06	1.204	8.44	0.074	0.029	2.23	2.333
12,000	7.549	0.920	0.20	0.051	0.06	1.231	8.78	0.074	0.029	2.23	2.333
12,500	7.996	0.950	0.20	0.054	0.06	1.264	9.26	0.074	0.029	2.23	2.333
13,000	8.453	0.950	0.20	0.057	0.06	1.267	9.72	0.074	0.029	2.23	2.333

[a] Assuming $0.08 barging charge.

the amount of fuel required to generate a given amount of electric energy not only because of the difference in heat content per ton but also because coals of the lower heat value usually cannot be burned at the same boiler efficiency due to losses caused by greater amounts of moisture and ash. The present price of oil, including fixed charges on incidental facilities required in a 300,000 kw plant operating at a 50 pct load factor, and operating expenses incident to its use, is about $2.33 per barrel. Table 7 shows the price that coal, containing various heat contents per ton, would have to be delivered to a San Francisco Bay area electric plant to make it comparable to oil delivered to the same plant at $2.23 per barrel. It shows that 22 million Btu per ton coal would have to sell for $6.79 per ton to be comparable to oil at $2.23 per barrel, whereas coal containing 26 million Btu per ton could be sold at $8.45 per ton and be comparable to oil at $2.23 per barrel.

Fig 4 shows graphically the estimated coal reserves of the various states by quality classification. The quantities shown in this columnar chart are approximately half of the amounts usually shown by the U. S. Geological Survey but are understood to represent the current opinion of Dr. Fieldner of the U. S. Bureau of Mines after allowance of a larger safety margin in the estimates themselves and in the mining losses.

There has been considerable controversy over the actual extent of our coal reserves which have been estimated from various geological observations, measurements of bed thicknesses, extent of surface outcroppings, trenching and exploratory tunnels, shafts and some drill cores made many years ago. There is even greater controversy over the amount of coal that can actually be recovered. Fig 4 shows 44 billion tons of bituminous coal in the state of Utah but it is doubtful, in the author's opinion, that more than 2 billion tons of coal comparable to that now being produced in the Carbon-Emery County fields, may be recovered by present methods.

Reappraisal Coal Reserves Necessary

There is great need for more accurate information concerning both the quantity and quality of solid fuel in this country—West as well as East—and much can be learned from accurate area surveys, diamond drilling, contour maps, and studies of surface geology. The National Bituminous Coal Advisory Council through its Coal Resources Committee, within the past two months, has transmitted to the Secretary of the Interior its report and recommendations concerning the need of and a plan to obtain a rapid reappraisal of coal resources east of the Mississippi River. A thorough and painstaking geologic and engineering survey of coal resources west of the Mississippi River should likewise be undertaken as rapidly as possible. The mountainous western terrain and the variability of western coal beds in extent, pitch, thickness, and quality will, no doubt, necessitate more painstaking study—requiring longer time and entailing relatively more expense—than the study of eastern fields.

Even though there appears to be much less commercially minable coal than we have been generally led to believe, there is undoubtedly sufficient coal in the Utah, Colorado, Wyoming, and New Mexico fields to supply all conceivable requirements for a period well beyond the interest of several future generations.

Table 8 shows the annual production from all Utah mines and those in the Rock Springs and Kemmerer, Wyo., areas for the 10 year period ending 1947. The 1947 production from Utah mines exceeded production during the war years whereas there has been a decline in production from the Rock Springs mines since the war due principally to greater use of diesel electric power by the Union Pacific Railroad.

An analysis of the potential coal production from Utah mines on a two shift full time basis, with consideration to available trained manpower and housing facilities, indicates that an additional 4 million tons per year could be produced to supply new West Coast markets. Similar available potential production from the Rock Springs and Kemmerer, Wyo., fields amounts to 2 million and 500,000 tons per year, respectively, making a total additional production capacity of 6½ million tons. Such productive capacity assumes absence of strikes, adequate railroad car supply, and a six-day week but makes allowance for ordinary mine disability and usual holidays. It should be pointed out here that such additional production could best be obtained by storing coal near the point of consumption during off-peak periods of domestic consumption—generally between early March and September. To emphasize

this point, Utah commercial coal mines showed the average monthly distribution (given in Table 9) of annual production for the 10 year prewar period ending 1939, when producton was not influenced by war or important contractual obligations for coal.

From data previously presented it is obvious that steam plants installed and under construction in central California will probably consume the equivalent of 3 million tons of coal per year by 1951 and that if the present program of meeting a large portion of required additional electric generating capacity by steam driven equipment continues, 6 million tons could be consumed in 1960. In dry years, with reduced generation from hydro facilities, fuel consumption would be considerably greater. Thus, the coal productive capacity necessary to supply all fuel requirements of steam electric plants now installed or that probably will be installed in central California by 1960 is already in existence in Utah and western Wyoming alone. Frankly this capacity is looking for a market and when found it can very easily be supplied with coal.

Reserves Should Be Assigned to Use

In order to outline some of the problems of the coal industry in supplying this potential demand, let us assume that an average of 3½ million tons of coal would be required per year for the generation of electric power in central California. Obviously the consumer of that quantity of fuel would desire to have blocked out and held in reserve coal for 25 years supply, or about 90 million tons; in other words that quantity must be assigned for specific consumption and it must not only be proved but must be amenable to economical mining.

Inasmuch as ownership or control of coal reserves is related to surface area of land, a discussion of this relationship seems in order. Nature has not deposited coal beds in a manner that will permit their complete extraction by presently known mining methods. In compiling inventories of coal reserves it has been the practice for some years to include all coal in beds above a minimum thickness of 14 in. at a depth not to exceed 3000 ft. Thus, in areas underlaid with multiple coal beds, generally the case in western coal fields, the reserves usually stated for that area include all coal in each bed above the arbitrary datum. In practice it has too often been found that removal of coal from a single bed will make extraction from others extremely hazardous, if not impossible. This is frequently true under careful mine development plans even when the uppermost minable bed is extracted first. Some of the factors affecting minability of multiple seams are distance between the beds, character of the intervening rock, and the amount and character of overburden.

Recovery Limited in Mountainous Region

In a given area of minable coal only a portion of the total may practically be extracted and in the western states an extraction of 60 pct of the total coal in any one bed is considered good practice by present methods. A figure often used assumes a yield of 1000 tons per acre foot and anticipates about 57 pct recovery of total coal, the balance being left as unrecoverable for various reasons. Conditions in many western mines require that some top coal be left to protect the roof. When bottom coal is deliberately left it is either to assure a firm floor for maneuvering equipment or to prevent the inclusion of impurities.

The minimum bed thickness required for economical mining of coal in the Rocky Mountain States is now considered to be about 4 ft but this will vary depending upon the character of the roof, value of the coal, and other conditions.

Mining experience in Carbon County, Utah, the Paonia region of western Colorado, and the Rock Springs area in Wyoming indicates that maximum recovery of perhaps 8000 tons per surface acre or 5 million tons per section (640 acres) may be assumed. Even where there is more than one minable bed the extraction per acre may not exceed this figure.

Table 8 . . . Annual Coal Production from All Utah Mines and Mines in Rock Springs and Kemmerer, Wyo., Areas 10 Year Period Ending 1947

Year	From All Utah Mines[a]	From Rock Springs, Wyo., Area	From Kemmerer, Wyo., Area
1938	2,946,951[b]	3,315,811[e]	426,490[e]
1939	3,284,904[b]	3,527,411[d]	449,002[d]
1940	3,575,586[b]	3,849,691[e]	452,228[e]
1941	4,076,779[b]	4,520,116[e]	486,297[e]
1942	5,516,849[b]	5,561,360[e]	580,988[e]
1943	6,781,298[c]	5,992,451[e]	616,622[e]
1944	7,206,107[c]	6,136,979[e]	543,966[e]
1945	6,738,462[c]	6,251,290[e]	500,884[e]
1946	6,166,410[c]	4,438,618[e]	395,537[e]
1947	7,619,378[c]	4,907,680[d]	420,636[d]
Totals	53,912,724	48,601,407	4,872,650

[a] Includes captive mines.
[b] Figures taken from Bureau of Mines records.
[c] Figures taken from Industrial Commission of Utah records.
[d] Figures taken from the Annual Report of the State Inspector of Coal Mines, Wyoming.
[e] Figures taken from Bureau of Mines records.

Table 9 . . . Average Monthly Distribution of Production from Utah Coal Mines

Month	Percentage of Annual Production
January	13.0
February	10.4
March	7.5
April	5.0
May	4.0
June	3.7
July	3.7
August	5.9
September	9.6
October	12.0
November	11.9
December	13.3
Total	100.0

Leasing Act Limits Controllable Reserves

Perhaps 70 pct of the coal reserves in the western states lie under Government land and the Mineral Leasing Act, as amended by the 80th Congress, permits no person, association, or corporation other than railroads operating as common carriers to take or hold coal leases from the Government, at any one time, exceeding 2560 acres (4 sections) in any one tract, or in the aggregate acreage 5120 acres (8 sections) in any one state. Under this act a single company cannot practically hold leases in Government land in any one state containing more than 40 million tons of recoverable coal. Therefore, under present law and presently known mining methods it would be possible for a single company to control 90 million tons (3½ million tons per year for 25 years) only by leasing Government land, known to be underlaid with coal, in three states.

At present day prices the cost of developing a mine in mountainous western areas may be assumed to be about $7.75 per ton of annual mine capacity, made up as shown in Table 10.

Table 10 . . . Cost of Developing a Mine in Western Area

Capital Expenditure	Estimated Cost per Ton of Annual Mine Capacity
Mine	$4.00
Tipple including preparation and washing equipment	1.00
Housing and community facilities	2.75
Total	$7.75

Thus, the assumed coal requirements of 3½ million tons for central California would represent an investment of about $27,000,000. Such an investment would require a firm commitment on the part of the buyer to enable the mining company to finance the improvements.

There has been a rapid increase in demand for slack coal (1⅝ in. by 0 and 1 in. by 0) and it often has to be prepared by crushing from mine-run sizes—it is no longer a byproduct. Consequently any slack coal produced for generation of electric power in California would have to carry the full overall cost of production as it could not be assumed to be subsidized by the larger sizes of lump coal.

Lower Mining Costs Doubtful

With such a high capital cost to provide productive capacity and, lacking any particular benefit of subsidy from the larger sizes of coal, it may well be asked what may be expected in the way of possible cost and price reductions with the use of newly developed mining machines so widely publicized during the past few months. More experience with this new machinery will be required before definite predictions can be made but the following appears certain:

1. The capital cost of new machinery will be relatively high.
2. Application to western mines will be limited.
3. Trend of mine labor costs will likely be upward.
4. Power consumption, and hence power costs, will likely increase.

These factors will possibly offset a large part, if not all, of savings that might be expected through use of this new and revolutionary mining machinery; any promise of a decrease in coal prices due to its employment seems dim at this time, but the possibility does exist. Such machinery will, no doubt, alleviate shortages of mine labor as well as the housing and community problems of coal mining towns and, in general, ease mine production problems. At the end of 1948 Utah slack coal sold at $4.60 per ton f.o.b. cars at the mine and it is the author's opinion that, since any coal for West Coast electric power generation must carry the full mine production expense, no price reduction can definitely be predicted to result from use of new mining machines.

Railroad Facilities Adequate

The coal regions of Utah, southwestern Wyoming, western Colorado, and northwestern New Mexico are within close proximity to main transcontinental railroads and, in most cases, are already connected to main lines. These gathering lines and their relationship to main lines, together with presently available trunk lines connecting the intermountain region with the West Coast, are shown by Fig 2. It will be noted that the Santa Fe runs between northwestern New Mexico and southern California points; the Union Pacific between Rock Springs and Los Angeles; the Southern Pacific from Ogden to California; Denver and Rio Grande Western from western Colorado and Utah to Salt Lake and Ogden; and the Western Pacific from Salt Lake to San Francisco.

Freight rates, at the end of 1948, on shipments of slack coal from the inter-

Table 11 . . . Freight Rates on Slack Coal to West Coast Points
In Effect December, 1948

Destination	Starting Point of Shipment					
	Rock Springs, Wyo.			Castle Gate, Utah		
	Distance Miles	Use: Marine Bunkers	Use: Domestic	Distance Miles	Use: Marine Bunkers	Use: Domestic
Seattle	1,152	$5.04	$5.35	1,269	$5.04	$5.65
Portland	974	5.04	5.15	1,093	5.04	5.45
San Francisco	975	5.04	5.95	1,024	5.04	5.95
Los Angeles	1,026	5.04	6.45	928	5.04	5.95
Earnings in Cents per Ton Mile on Above Freight Rates						
Seattle	1,152	0.437	0.464	1,269	0.398	0.446
Portland	974	0.518	0.529	1,093	0.460	0.497
San Francisco	975	0.517	0.610	1,024	0.493	0.580
Los Angeles	1,026	0.491	0.630	928	0.543	0.642

mountain region to West Coast points for domestic consumption and for export are shown in Table 11. There is considerable disparity in rates from both Rock Springs, Wyo., and Castle Gate, Utah, to the four coast cities where the slack coal is to be used for purposes other than export. The rate on coal to be exported is the same from either starting point to any of the four coast cities even though there is a difference of as much as 341 miles in the shipping distance.

It is interesting to note that the freight rate between Sunnyside, Utah, and Fontana, Calif., on coking coal is $5.05 per ton and that coal up to 8 in. can be moved on this rate if it is suitable for coking. This rate was published late in 1942 on a contemplated annual movement of more than 500,000 tons.

There have been decreases in freight rates since 1923 on movements of slack coal from Utah into Seattle and Portland due to pressure on the railroads and to greater quantity of coal shipped.

It is the author's opinion that a movement of slack coal in excess of 3 million tons of coal per year from Utah to any point in central California would justify a freight rate equal to that published for Fontana, Calif., or $5.05 per ton.

The movement of 3½ million tons of coal per year on the basis of 240 mine working days per year would require that 14,600 tons be handled each mine working day. If it is assumed that shipments could be arranged for a 6-day week, the average railway movement would be 11,200 tons, or approximately 3 trains containing fifty-five 70-ton coal cars per day.

Such movements of coal would require railroad equipment represented by the investment amounts stated in Table 12 and entail the services of 250 men.

Table 12 . . . Railroad Equipment and Investment Required To Move 11,200 Tons of Coal a Day from Utah to California

Railroad Equipment	Cost
2800, 70-ton coal cars at $6,000 each	$16,800,000
32 locomotives at $310,000 each.	9,920,000
Miscellaneous equipment.........	5,000,000
Total.......................	$31,720,000

It therefore appears that, under presently known mining methods, the lowest price at which coal could be sold f.o.b. the mine in amounts of 3½ million tons per year for generation of power in central California would be $4.60. It also appears that the lowest freight rate that could be expected between intermountain points and the central California area would be $5.05 per ton, making a total cost of coal delivered at a plant site of $9.65 per ton.

Conclusions

The following are the author's conclusions:

1. Coal mines in Utah and in the Kemmerer and Rock Springs districts of Wyoming could increase annual production by 6½ million tons per year.
2. Under present conditions coal could probably be delivered to any steam electric plant in central California at a price not to exceed $9.65 per ton.
3. The use of coal at such a price, while higher than the equivalent present price of fuel oil, is entirely feasible.
4. There are adequate railroad facilities for movements of large quantities of coal from the intermountain region to the West Coast.
5. In a national emergency it probably would be extremely difficult, if not impossible, to obtain sufficient oil to meet requirements of the greatly expanded West Coast steam electric generating capacity.
6. The intermountain region contains ample coal reserves to supply all conceivable demands for West Coast power generation for a number of generations.
7. Increasing demands of labor threaten to lessen, if not eliminate, savings in cost of coal production through the use of new mining machinery.
8. Continuation of experiments in socialism by the Federal Government through construction of hydroelectric generating plants, particularly those unrelated to land-use reclamation, defies justification. Rates under this concept of a governmental function are subsidized through greater taxation of its people. Private capital is available to construct steam plants, or hydro plants where feasible, and should be permitted to continue in order to preserve the principles of our free enterprise system.

DISCUSSION

(*L. C. McCabe and Robert P. Koenig, presiding*)

C. G. Ball*—I was asked to lead off the discussion, but it is not with the thought that I might be able to add anything to the paper. The thoroughness with which it was prepared rather forestalls the asking of many questions. Your treatment, Mr. Heiner, is a very valuable contribution. I do want to suggest—that although you have limited your study to this specific question, with certain geographic limitations, many of the things in your paper apply just as well to the eastern coals. I want to agree 100 pct with your final conclusion concerning government-subsidized construction.

* Paul Weir Co., Chicago, Ill.

L. C. McCabe*—It is certainly worthwhile to take stock occasionally to see where we are going in problems of this nature. I agree with Mr. Heiner that ultimately the only reliable source of fuel that the West Coast has is coal but the time factor is the difficult element to evaluate.

Just before I came here I discussed this subject with N. B. Hinson, Vice President and Executive Engineer of the Southern California Edison Company, and Chairman of the West Coast Inter-Power Exchange Committee. He has given much thought to utilities' fuel supply and it was very helpful to me in preparing a discussion of the paper to talk with him beforehand.

Stock taking and forecasting of future development are essential to the continuing success of any enterprise. Mr. Heiner has called attention to the unprecedented growth of central and southern California and to the increased demands for fuel and power which have accompanied it. He discusses the increased fuel oil and natural gas requirements and the probable limits on the future use of these fuels and of hydroelectric power. In contrast to the calculable limits of these sources of electric energy, the author points to the availability of enormous reserves of coal in the Rocky Mountain States adjacent to the Pacific Coast which can be utilized for power generation. That there will be increased use of coal for power generation in the area under discussion is generally accepted but it is in the timetable of such development that there is not complete agreement.

In a recent report, Mr. Hinson reviewed the future power outlook for the Pacific southwest area. He pointed out that the use of steam plants in connection with water power plants in this region has made the maximum use of hydroelectric energy possible, and that the correct balance between hydro and steam generating plants produces the most economical overall system. Steam plants in the area which had been installed to protect against deficiency in hydro energy in dry years were used to carry war loads. Fortunately, no dry

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years occurred during the war so that in spite of the many new industries which moved into the territory, practically all demands were met. The construction of housing as well as the expansion of industry were major activities after the war. All of this activity created more load which made additional capacity necessary. Due to the shorter time of getting steam plants in service, the greater part of the additional capacity is provided by steam.

Mr. Hinson reported that the present installed capacity of the Pacific southwest power area at the end of 1947 was:

Hydro	3,300,000 kw
Steam	1,800,000 kw
Total	5,100,000 kw

The additions on order and under construction in the area for the years 1948, 1949, and 1950, amount to approximately 930,000 kw of hydro, and 1,050,000 kw of steam, or a grand total of additions of 1,980,000 kw. By the end of 1950 the total installed capacity will be:

Hydro	4,230,000 kw
Steam	2,850,000 kw
Total	7,080,000 kw

These additions are scheduled as follows:

Year	Kilowatts
1948	1,106,000
1949	163,000
1950	700,000

It is estimated, discounting war peaks and the present high rate of growth, that the electric load will double in 18 or 20 years, or by approximately 1965. Mr. Hinson indicated that on the Colorado within economic transmission distance of southern California, are located the proposed Bridge and Glen Canyon hydroelectric projects. These two sites have a capacity of approximately 1,300,000 kw as shown by some of the preliminary studies. There is approximately 400,000 kw of hydroelectric capacity that can be developed in southern California and approximately 1,000,000 kw of generating capacity in northern California. To supply the 3,500,000 kw of additional generating capacity will require an additional 800,000 kw of steam after all available hydro is developed in the area. This additional steam capacity with that now in existence and under construction would make a total of approximately 3,500,000 kw.

It is concluded that more steam capacity may be necessary to provide adequate protection against low water years and that this increased amount of steam capacity will require large quantities of fuel. As Mr. Heiner points out, all of these steam plants are designed to burn oil or gas, and the more modern ones are designed so that with a minimum of change in the boilers and with the addition of special equipment, they could use pulverized coal.

The study for the Pacific Southwest Power Interchange Committee concludes that when the load in the area has doubled, practically all the additional generating capacity needed to carry the new load from that time on will have to be supplied by fuel burning plants. The possible use of atomic energy in the production of electric energy is discussed briefly. It is pointed out that if this form of energy should be used in modern steam plants, it will only be necessary to replace the existing boilers and fuel-burning equipment by atomic piles and their auxiliaries producing steam. This would mean replacements valued at 25 or 30 pct of the present investment in existing steam plants.

The prediction of declining petroleum production in the '20's and the subsequent expansion of that industry, suggest caution in estimating production potential of the petroleum industry. Within the past two weeks the price of fuel oil for power generation has declined 20 cents per barrel. Whether this trend will continue over a period of years cannot be foreseen but it does emphasize the present state of uncertainty in the whole fuel market.

It may be carrying speculation a little far to anticipate any appreciable use of fissionable minerals for power in the immediate future but it should not be ignored.

I note that in a paper given yesterday in the Oil and Gas Division, it was pointed out that the recent dieselization of railroads on the West Coast is a material factor in the fuel oil problem. It was stated that a diesel locomotive requires between a fourth and a third of the amount of oil that oil-burning locomotives consume. This has made more fuel oil available quite recently and within the past few weeks has contributed to the upset of the fuel oil market on the West Coast. That is a short-term trend, perhaps, but it is a factor and one that needs to be kept in mind in any study of the market position of the fuels.

Mr. Heiner has contributed a valuable paper which will be stimulating to the coal industry of the Rocky Mountain States and the power industry of the Pacific Coast. Studies of this nature are essential in planning the development of industry to serve the country in time of peace or national emergency.

C. P. Heiner (author's reply)—Mr. McCabe mentions the possible development of the Bridge and Glen Canyon hydroelectric projects by the government, having a potential capacity of 1,300,000 kw. In my opinion, the construction of such projects—unrelated to land use reclamation—cannot be justified, as electric rates under this concept of a government function are subsidized through taxation. Private capital is available to construct the necessary electric capacity, either steam driven or hydraulic driven, where feasible, and should not, by government action, be precluded from doing so.

V. F. Parry*—I found this paper extremely interesting and very thorough. The author and his staff are to be congratulated on this very thoughtful piece of work. From my point of view it seems conservative. The figures that Mr. Heiner uses are quite safe, I would say. A study of trends of electric power for the United States as a whole indicates that there is approximately a 50 pct increase each ten years. The author shows a 70 pct increase for the West Coast area, which is entirely consistent with the increase in population. The planning in reserve coal that would have to be blocked out for such power generation is necessarily affected by the increased power consumption.

In going over the paper, I noticed that gross Btu is used for comparison of energy in the form of coal and oil. I believe that the use of the net heating value would show a more favorable picture for coal. For instance, in well-engineered plants, the net heating value in relation between fuels can be obtained fairly accurately, and if this method of comparing fuels on a net heating value basis were used, the price of oil, calculated in the paper as $2.62, compared with $9.65 for coal, becomes about $2.50. It decreases the spread of 18 pct that now exists and I believe will make a more favorable picture for coal.

I was glad to have the author mention the need for storage of coal, because that has been foreseen as one of the only solutions of bringing coal to the West Coast. The load factor on coal produced in the Rocky Mountains varies in a ratio of about 3½ to 1 between summer and winter months. If large quantities of coal were shipped here, it would require storage, which can be done. It would also alleviate the intermittent mining in Utah and Colorado which, I believe, would tend to cut the cost of coal considerably.

Eugene McAuliffe†—Next to food, fuel, whether coal or liquid, is the foundation stone of western world economy. To this end we should not fail to maintain a strong and readily expansible fuel industry. Mr. Heiner has shown the limitations of our western hydroelectric source of energy. Such should be expanded, where it is economically possible to do so, but I am not sure that he has sufficiently stressed the blighting effect of an extended cyclical drought, that would cut deeply into the supply of hydroelectric power.

As Mr. Heiner has suggested, the coal production of the west has suffered from the rapid substitution of diesel for coal-burning locomotives and that impair-

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ment is steadily going forward. Without attempting to "view with alarm," I am wondering where the western railroads and industrial users of fuel oil for steam making purposes would get a substitute coal supply if war, or a threat of war, should result in a government mandate to cease the use of oil for making steam.

What the safety of the fuel supply of the West most needs is a strong coal industry, with the greatest possible substitution of machinery for hand labor and, in many instances, the development of improved housing near the mines, with other props, such as adequate school and recreation facilities, that will stabilize mine labor insofar as possible. We must keep in mind that our western railroads, largely depend on single track movement of freight and passengers, which tends to slow freight traffic, making added coal cars necessary. The railroads, under the pressure of oil competition, have made heavy investment in tank cars, with the result that for the first time in years, coal cars have been very scarce. Perhaps it is time to begin to buy gondolas rather than tank cars. The storage of coal at destination should be developed and encouraged, and somehow coal mining labor and their leaders must be shown that when our national economy needs fuel, it must be mined and moved without interruption.

C. P. Heiner—Mr. McAuliffe has presented some very worth-while and arresting reflections. First, a cyclical drought would have a crippling effect on high load factor industries dependent on hydroelectric power. It should also be noted that hydroelectric plants are no better than the water supply they depend upon. The simple expedient of building dams and hydro plants will not guarantee the supply of power to the vital industrial plants of the West. More and more steam plants must and will be built to make good the inevitable shortages of power produced by hydro plants. This applies to the Pacific Northwest as well as to California.

Second; I agree with Mr. McAuliffe that a healthy coal industry is our best industrial insurance policy in case of war or threat of war. A national policy of oil and gas conservation would aid in maintaining a healthy coal industry.

Third; information is at hand to indicate the railroads now have sufficient motive power and gondola cars to move at least 2,000,000 tons per year from Utah and Wyoming mines to the West Coast. Thus, both transportation facilities and mine capacity are ready to permit immediate extensive use of coal in large West Coast steam plants.

G. H. Cady*—This paper is very interesting. The author has presented a good case for the Utah-Wyoming coal industry as a source of fuel for the California utilities. I suppose the most critical consideration is the matter of reserves as discussed on pages 391 and 394 to 396 including the rate of recovery.

It is generally acknowledged that the present data on reserves is unsatisfactory. I think it is particularly so when presented in the form of maps. These suggest that the quantity of coal in various areas is indicated by the relative sizes of the areas. In general it is much easier to map the extent of the "coal measures," that is, groups or formation containing coal beds, than to map the actual extent of the area underlain by workable coal beds. For example, the area of the Eastern Interior Province in Illinois, Indiana, and Kentucky in which the Pennsylvanian "coal measures" are present can be readily indicated. It is much more difficult to indicate the area underlain by workable beds, which is considerably smaller.

Furthermore the idea of what constitutes a workable coal bed is changing from year to year. Forty years ago a 3 ft bed would have been regarded as unquestionably workable; hence the use of 14 in. as a minimum thickness of workability. This thickness did not seem unreasonable as it may today. Mining in a 3 ft bed on a commercial scale seems to be a thing of the past. In fact, in certain regions—the Middle West and the Rocky Mountain region—the limit seems to be about 4 ft.

To return to the matter of maps: What is true of Illinois, Indiana, and Kentucky I suspect is true of the western states, that is, areas mapped represent the extent of the Cretaceous and Tertiary "coal measures" and are therefore, too large to represent the actual extent of workable coal beds, which have not yet been accurately mapped. It is probable that the present tendency would be to mark off as very doubtful "reserve" the areas which are designated on the author's maps as "doubtful" that is with broken-line patterns. One wonders what conditions exist that make these areas doubtful.

Reappraisal of the coal reserves (p. 394) is certainly desirable but reappraisal is possible only on the basis of facts relative to the occurrence, distribution, and characteristics affecting mining of the different beds. Somehow there seems to be a feeling that the mere acknowledgment of the desirability of reappraisal will do the job. As a matter of fact, in many areas information about the coal resources is little if any better than it was forty years ago and the guess at that time would be matched by a guess today. The results, if different, would be so mainly because different criteria of workability were used. What is needed is a resumption of the coal-resource surveys that were carried on by the Federal government. These were largely abandoned before the war.

In regard to recovery (p. 395), I am surprised that the author's recovery is as good as reported. The estimate of 8000 tons per acre of surface area is, I believe, better than prevails in Illinois even in Franklin County where 7000 tons is the usual figure given by engineers, and it will not do to examine *that* figure too closely. This is, of course, all room-and-pillar mining by which method 99 pct of our coal is now extracted.

Our mining people seem to think that the new continuous mining machines are going to answer some of their problems but the items mentioned on p. 396, except possibly No. 2 seem to be valid. From what I have seen in Illinois I would say that success in their use depends very largely in having them work in pairs and continuously. They do a wonderful job. Judgment on the value of the machines will have to be deferred until they have passed the experimental stage.

I shall be interested to hear the reaction to this paper, the main thesis of which seems to be that Utah and Wyoming provide the logical sources of solid fuel for the California utilities, and as such, deserve certain concessions in regard to coal land ownership, control, and freight rates. It appears logical that if the railroads can afford to move export coal and coking coal for $5.05, they can afford to move other coal. I do not know how much of an issue will be made of the reserves problem, but it seems to me a rather critical one and that 25 years is altogether too short a time to look ahead—or is the author uncertain about this atomic energy business?

C. P. Heiner—Dr. Cady brings out the need of stressing the importance of considering the areas underlain by workable coal beds rather than the areas containing simply "coal measures." There is a great deal of so-called "coal land" in Utah and other intermountain states containing coal measures that likely can never justify mining operations. There are many miles of poor coal outcrops within the so-called "coal lands" and much exploration and geologizing must be done to determine the extent of any recoverable coal beds in such land.

In Carbon County, Utah, where the pitch of the coal beds is often 8 to 10 pct, a few mines have operated in 40 in. of coal. Frequent splits and/or areas of poor roof make such operations extremely costly so that a 4 ft bed is generally assumed to be the practical low limit of thickness. This same condition is also known to exist in Colorado.

I heartily agree with Dr. Cady that any reappraisal of western coal reserves should mean a resumption of coal-resource surveys that were formerly carried on under the U. S. Bureau of Mines. These surveys will also likely require much systematic core drilling some distance back from the coal outcrops.

Dr. Cady raises the question as to the propriety of assigning only 25 years of coal reserves to steam generating plants.

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That period was assumed as an absolute minimum life for modern steam plants. Government imposed acreage limitations make it difficult to assign at the outset a 25 year supply of coal for a large steam plant.

I am not too much impressed by the threat of atomic power plants doing away with the value or need of coal reserves.

M. G. Bluth*—The author has presented some very interesting and original data, opinions, and observations on a subject which has had little if any publicity, and which has extremely important implications in the economy of the western parts of the United States during periods of peace in the years ahead. In an emergency of national and international scope, the author's analysis would bear close study by both industry and Government authorities charged with maintenance of protection of the West Coast and the necessity of making power available for production of essential materiel in time of military operations.

The author has assumed that the fast growth in population in California will continue for some years to come. This appears to be a reasonable and fair assumption when the record for the past 20 years is examined—and especially since the start of World War II, during and since the end of that war. The population has increased in California to an amazing extent in the past few years and population increases of this proportion create and accelerate the growth of industry and commerce. The obvious demand, then, for power for industry as well as domestic use, to serve the growing industrial areas of California, means that fuel availability, water resources, and other physical elements in the generation of power and electricity stand as most important factors in order to meet any and all requirements under such a trend.

The paper did not emphasize, or perhaps to state it in a more direct way, it did not point out too clearly the need for extraordinary storage capacity to take care of the huge reserves of coal on the Pacific Coast (California) which would be most necessary and vital in periods of extreme national emergencies, as well as to safeguard against strikes of miners, railroad and transportation strikes, and acts of God. Both standby plants and extra heavy storage piles of coal would be necessary at plant sites in order to overcome the handicap of distance and other natural barriers. It seems prudent, under world conditions of unrest, to "play safe" in providing both the storage facilities and the standby plants, particularly in strategic locations where industry and the safety of workers and citizens are so important to the economy as well as assuring productivity in times of war.

While the author states that, in the interest of simplification his discussion is confined to the central part of California, it would be desirable to devote considerable study and attention to southern California. This region is enjoying a much faster rate of growth in population and is becoming increasingly more important as a center of industrial activity. Because this region was not included in the paper, questions naturally arise as to what major differences exist between central and southern California power requirements and problems; and if coal should prove to be essential to power generation on the Pacific Coast, what are the physical factors prevalent in southern California as compared to other sections of the state?

The author presents a brief diagnosis of Government hydroelectric power projects in the early part of the discussion. More emphasis could have been placed on, and the fact stressed that, Government projects are tax-free and as such private power enterprises cannot compete. Government hydroelectric power projects, on the other hand, could not compete with privately operated power companies if such Government plants paid their full share of taxes.

In one paragraph the author mentions the Colorado River power possibilities. His contention could have been strengthened if he had said that in addition to the high cost of constructing transmission facilities from the river to the Coast, the cost of transmitting power in such long distances is very high and the power losses are great—at least in the present stage of the art and development of transmission lines.

The author briefly covers the importation of gas and oil into California. Reference could have been made to the fact that utilities receive a much higher rate for gas sold to domestic consumers. Gas utilities, generally, take on large industrial and process gas users to provide an outlet in off-peak periods at dump load rates and as soon as they are able to build up a volume of business and demand for gas from domestic users, they soon drop their interest in industrial users except at the usual noninterruptible rates. Under such procedure, gas is not a practical fuel for year-round use in electric generating stations, but some uses can be provided for gas in such stations during off-peak periods and perhaps as a standby fuel in periods of emergency when the regular fuel is not immediately available.

No attempt has been made to analyze the portions of the author's presentation relating to coal reserves, mining costs, production, and other related matters. The particular part of the problem under discussion appears to be adequately covered pending more detailed and penetrating surveys which probably would be necessary in the event of a sudden demand or need for large quantities of western mined coal on the Pacific Coast.

In planning for national defense and military alertness, it is essential that the coal reserves and production possibilities and potentialities of the coal bearing and producing states of the Far West be given the careful attention of Government authorities and private enterprise, as one of the important factors in making our Pacific shores a strong bastion of both protection to the nation and the economic welfare of the national community.

The author has made an important contribution to a better understanding of the economic problems and fuel problems in the West. The paper, on the whole, should receive widespread attention and careful study of all those engaged in the field of power, national defense, and economic stability.

* Manager, Chicago Office, National Coal Association, and Executive Secretary Stoker Manufacturers Association, Chicago, Ill

C. P. Heiner—The author stated in the paper that to deliver coal at the lowest possible cost on the West Coast, advantage should be taken of low mine production during the non-heating season which would entail storage of coal at points of consumption during those months. He does not agree with Mr. Bluth that any more coal would have to be stored on the West Coast during a national emergency or for protection against strikes than at any other part of the country. The author arrives at this conclusion because of the availability of several railroads between mines and points of consumption. He also does not agree that any more standby generating plants would be needed on the West Coast than any other coastal regions of the United States.

The author agrees with Mr. Bluth in regard to construction by the government of certain hydroelectric projects. He firmly believes in western land reclamation and, where generation of electric power is a proper adjunct to land-use reclamation, he believes that the most feasible method of power generation would be for private industry to construct the generating plants at the government dams, pay for the use of falling water and pay taxes on its operation. He does not agree with the concept that the government should construct generating plants and transmission lines on practically an interest and tax-free basis and compete with private, tax-paying free enterprise as such a concept is socialistic.

The author agrees with Mr. Bluth that even though large quantities of natural gas will be imported into California, its availability for continuous industrial use will probably decline not necessarily because of the higher monetary yields from domestic and commercial customers—such customer classifications consume gas at lower load factors than industry—but because of probable future actions of regulatory bodies.

Discussion*

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* TP 2695 ABFH.

A—Metal Mining

Alluvial Tin Mining in Malaya

By A. D. HUGHES, Member AIME

DISCUSSION

(*C. M. Romanowitz and K. Fritz Eilers, presiding*)

C. W. MERRILL*—Mr. Hughes' paper not only is very well presented but is most timely in that it covers a subject of vital interest to the United States. Tin is one of the strategic metals which has not been found here in appreciable quantities. The total United States tin output since the first recorded production over 100 years ago would not supply current consumption for a fortnight. On the consumption side, tin is extremely important in national defense because of its use in tin plate which, in the form of tin cans, makes it possible to feed properly the armies that the United States recruits from its civilian population in emergencies. Moreover, the internal combustion engines so important in powering tanks, planes, and other war machines, require large quantities of tin alloys for bearings and for soldering. These considerations, along with the fact that Malaya, historically, has been the leading source of tin to the United States, make the kind of information that Mr. Hughes has presented so ably, a matter of vital interest.

Your Chairman, Mr. Romanowitz, in his introduction, referred to the fact that I had been a member of the United States delegations to the three meetings of the International Tin Study Group. Some comments on the Group's objectives and deliberations may be of interest.

When it appeared that the International Tin Committee, the prewar producer's cartel, might soon revive its operations and extend its control agreement beyond the expiration date of Dec. 31, 1946, the United States Government became interested in some kind of an international trade organization concerned with tin which would give the consumers' interest equal representation with the producers' interests. An international meeting was called in London in October 1946 to consider this problem. I did not attend that meeting but the result of the meeting was the organization of the International Tin Study Group which

* U. S. Bureau of Mines, Washington, D. C.

called its first meeting in Brussels for April 1947, which I attended as a member of the United States delegation. At that meeting the tin supply situation was studied and a permanent Tin Study Group Secretariat was agreed upon to be headquartered in the Hague.

A second meeting of the Study Group was called in April 1948 in Washington, D.C., where again I attended as a member of the United States delegation. This meeting reviewed the tin supply situation. Among other items placed before the Group was a request by one of the delegations that the Tin Study Group proceed immediately to determine the advisability of an international conference to negotiate a commodity agreement under the International Trade Organization Charter. A working party met in the Hague in June 1948 to consider this problem.

A third meeting of the Tin Study Group was called for October 1948 at the Hague where again I was a member of the United States delegation. The report of the working group was considered. The Tin Study Group requested its delegations to report back to their respective Governments on the Group's deliberations and to request their Governments to inform the Tin Study Group Secretariat whether the negotiation of a commodity agreement for tin under the Havana Charter of the International Trade Organization was appropriate at this time. The Hague deliberations of the Study Group had developed a tentative framework for such negotiations. The reports by the various Governments as to their wishes with regard to such negotiation are now being received by the Secretariat. The closing date for reporting is now some two weeks off.

Drilling Blastholes at the Holden Mine with Percussion Drills and Tungsten Carbide Bits

By ELTON A. YOUNGBERG, Member AIME

DISCUSSION

(Guy N. Bjorge and Lynn Hersey, presiding)

J. H. HEARDING, JR.*—Extremely hard ferruginous chert (taconite) was encountered in driving a drift at the Fraser underground mine near Chisholm, Minn., on the Mesabi Iron Range. In order to get better results in drilling in this ground, it was decided to try tungsten carbide bits.

In the beginning, 1½ in. tungsten carbide bits and 1⅛ in. round-lugged alloy steel drill rods with "jack-stud" inserts were used. Considerable difficulty was encountered because the jack-stud became loose in the drill rods. Because there was not enough alloy steel drill rod on hand, we began to insert jack-studs in the ordinary high carbon drill rods and found, to our surprise, that there was less trouble with jack-studs working loose in these rods than there had been with the alloy steel rods. As a result, use of the alloy steel rods has been discontinued entirely.

Under the heading "Drilling Procedures" the statement is made that all holes are started with detachable bits or conventional steel to prevent undue strain on the tungsten carbide bit inserts when collaring a hole on uneven ground. This practice was followed in our work originally for the same reason. It was soon found, however, that the holes could be started with the tungsten carbide without injuring the inserts if the drill operator was careful and did not give the drill too much air until the bit had a chance to seat itself in the rock.

* General Superintendent, Oliver Iron Mining Co., Hibbing, Minn.

J. C. FRANZ*—In a discussion of other technical papers with reference to percussion drills and the use of carbide tungsten bits, it was my impression that the authors reported one of the first difficulties experienced with the use of standard, conventional percussion drills, was insert bit failures due to crushing of the bit inserts when the bits became dull.

During the past few years, some of the rock drill manufacturers, I am told, have adopted an air drill for use with the carbide bits that have a lighter blow and faster rotation than the regular machine. This machine has been tested and the carbide bit failures have been considerably reduced.

L. W. DUPUY†—The difficulties Mr. Youngberg mentions that were had with rod and coupling breakage call to mind similar troubles experienced at the Picacho mine in California near Yuma.[6] The commercial couplings then available were found to be tempered almost to brittleness. Rod breakage was usually in the threads near the end of the rod. The highly-tempered couplings would break as soon as the rod broke. The solution was to thoroughly anneal the couplings and to put in only a very mild temper. Subsequently the alloy used in making the couplings was changed from the original by the manufacturer, but still only a very mild temper was used. The result was that the softer coupling would give but would not snap and release its hold on the broken rod thread. Fishing jobs were reduced to the few where the rods broke at the rod end of the thread. Thread breakage on the rods was further reduced by thorough annealing prior to tempering.

The data Mr. Youngberg presents regarding the use of carbide inserts on long blasthole drilling is most interesting and it would appear that the use in this manner of percussion drills and carbide bits will have many applications.

* Division of Industrial Safety, Department of Industrial Relations, State of California, San Francisco, Calif.

† U. S. Bureau of Mines, Rolla, Mo.

[6] Leon W. Dupuy: Sampling the Picacho with Drill and Vacuum Collector. *Eng. and Min. Jnl.* (Jan. 1940).

W. M. WOODWARD*—How many feet of hole, on the average, do you get from a tungsten carbide bit, and is there much variation from this average?

E. A. YOUNGBERG (author's reply)—In recent months the 2 in. tungsten carbide bit footages in long hole drilling have varied from 115 to 140 ft monthly. The footages obtained from individual bits vary from several feet to over 300 ft, however the life of a majority of the bits will be within 25 ft of the average.

* The Cananea Consolidated Copper Company, S. A., Cananea, Son., Mexico.

Diamond Drilling Quartz-feldspar Intergrowths

By L. C. ARMSTRONG, Member AIME

DISCUSSION

(*Guy N. Bjorge and Lynn Hersey, presiding*)

A. E. ROSS*—Mr. Armstrong in his paper stated that they had experienced considerable difficulty in drilling the quartz-feldspar intergrowths. The diamond loss was excessive and the diamond bits polished quite rapidly.

Mr. Davidson, who presented the paper, stated that bortz stones in the range of 20 to 30 per carat had been used in the bits on this job. I would like to suggest the use of bits employing stones in the range of 60 to 100 per carat, an average of possibly 80 per carat. If these stones had been used the bits might have lasted longer and not polished so quickly.

As a basis for this suggestion I would like to point out that in northern New York State we had at least two examples where bits employing stones in the range of 80 per carat had solved a difficult problem. Bits set with stones in the range of 20 to 30 per carat had been used previously and had failed. With the finer stones, the cost per foot of drilling decreased materially.

It should also be noted from a technical standpoint that the smaller stones require higher drilling speeds than the larger stones. Greater pressures are not normally necessary, and it is the higher speed which seems to be the major factor of importance.

It should also be noted that the finer stones afford a greater number of cutting surfaces for the same carat weight in a bit. Then too, the smaller stones, by virtue of their size, serve as sharper cutting edges than larger stones. There is no doubt that the diamond bit employing smaller stones costs more to set than the bit using larger stones. However, with the technique developed at the shops of Sprague and Henwood, Inc., we feel that the relatively slight additional cost required to set these bits is more than paid for by the resultant lower drilling cost. We feel that the highest quality stone available is the most economical to use when hard fine grained rock is encountered.

In summary, I would suggest that it might be possible to solve the problem presented in this paper by using a diamond bit employing stones at least as small as 80 per carat of the highest grade possible and by using the diamond bit at the highest speed available with modern drilling equipment.

L. C. ARMSTRONG (author's reply) —It is generally agreed that the use of smaller, high-grade diamonds and of higher drilling speeds, as suggested by Mr. Ross, may be the best approach to the solution of the problem.

* Assistant to President, Sprague and Henwood, Inc., Scranton, Pa.

Safety Practices at the Crestmore Mine of the Riverside Cement Company

By R. H. WIGHTMAN and G. H. ADAMS, Members AIME

DISCUSSION

(*L. A. Walker and H. C. Weed, presiding*)

H. C. WEED*—Referring to the use of "dummy fuse" for checking the shots in chute blasting operations, I believe that an even better practice is to blast the chutes with no delay electric blasting caps.

Permanent wires can be strung through the grizzly lines and the lead wires from the shots attached to these lines. Using this method, no time is lost in spitting the individual fuses. The possibility of a fused charge becoming dislodged by previous shots and falling into the chute below is eliminated and there is no chance of a man walking back into his own blast.

A small hand twist blasting machine or a radio "B" battery can be used to fire the shots if the lines are fired individually. If so desired the entire area can be blasted at once by a switch at the manway or in the drift below, providing a sufficient source of electricity is available. In case a switch is used, proper precaution for locking the switch must be provided.

R. H. WIGHTMAN (author's reply) —We do use electric blasting when the number of shots is large; but we have found it more satisfactory to use fuse and caps for individual shots on the grizzly.

* Inspiration Consolidated Copper Co., Inspiration, Arizona.

B—Minerals Beneficiation

Effects of Rod Mill Speed at Tennessee Copper Company

By J. F. MYERS and F. M. LEWIS, Members AIME

DISCUSSION

C. G. McLACHLAN*—I have read this paper with considerable interest and wish to congratulate the authors on the care with which they carried out their experiments and for the detailed sizing data they have presented. On the other hand I do not feel that these data establish their contention that "with other factors the same, the work accomplished in a fine crushing rod mill is directly proportional to the rotating speed."

My reason for making this statement is that their conclusion is based on the percentage of minus 65 mesh material produced at each of the speeds at which their tests were run.

The rod mill is, however, not a fine grinding machine and therefore figures based on minus 65 mesh production are figures outside the range at which it operates to best advantage. Further, if rod mill performance is to be judged on the production of such material it should have been established that a rod mill will produce it as effectively as a ball mill.

* Noranda Mines Ltd., Noranda, Que., Canada.

To check this last point I have gone over some of the rod and ball mill grinding figures for our Waite-Amulet operation and find that the ball mill at that property produces more minus 65 mesh material per horsepower than the rod mill. Moreover, this comparison is even more favorable to the ball mill if it is limited to minus 100 mesh production.

It therefore seems to me that the most satisfactory basis on which to try to compare the performance of the rod mill in the present case is by taking a weighted average of all the new surface produced per horsepower.

W. H. Coghill[3] has published data which can be applied to the sizing figures presented by the authors, and if this is done the units of new surface per horsepower produced in the Tennessee tests are as follows:

Critical speed, pct.	82.4	74.6	70.5	66.5
Units of surface produced per hp.	1.000	1.014	1.023	1.027

[3] W. H. Coghill: Evaluating Grinding Efficiency by Graphical Methods. *Eng. and Min. Jnl.* (1928) 126, (24) 934.

Admittedly the increment of increase in new surface per horsepower shown in the foregoing figures is not great, but its trend is nevertheless definitely in favor of slower speed operation, for the conditions under which these particular tests were run.

I also feel that some information regarding liner contour should have been given as I think it will be found that a mill which is equipped with liners which lift the rod load can be run at a slower speed than a mill in which the liners are comparatively smooth.

J. F. MYERS and F. M. LEWIS (authors' reply)—The position taken by Mr. McLachlan is to be respected even though we are all aware of the faults of surface considerations. As he points out, the increment of increase is very small. It is so small in fact that for all practical purposes it does not conflict with the conclusions of the paper. The authors appreciate Mr. McLachlan's comprehensive study of the data.

Humphreys Spiral Concentration on Mesabi Range Ores

By WHITMAN E. BROWN, Junior Member AIME, and LOUIS J. ERCK

DISCUSSION

(*I. M. LeBaron and F. R. Milliken, presiding*)

L. A. ROE*—This paper is one of great value to the iron ore industry. The Humphreys spiral is a relatively new tool and gives promise of being quite useful in solving certain problems of iron ore beneficiation. Spirals have been tested on a martite ore at the Benson mine of Jones and Laughlin Steel Corp. in northern New York State. Our New York ore is considerably different, physically and mineralogically, than the one mentioned in this paper, and contains only 25 pct iron. This ore can be processed on spirals to give a concentrate containing 62 pct iron with an 85 pct recovery. On our particular ore we found operating difficulties when the spiral feed was coarser than 14 mesh. These were chiefly due to excessive "build-up" of locked middlings in the spiral circuit. In extreme cases these middling particles would accumulate to such an extent that the plant had to be shut down and the spiral surge tanks cleaned out.

It is interesting to note that there exists a close relationship between the results of tabling a given iron ore and concentration of this same ore on Humphreys spirals. The size range of the ore must, of course, be within those limits acceptable to spiral concentration. Comparative tests on several of our martite ores showed tabling results to be the same as spiral results.

The authors make no mention of the use of a tailing stream splitter (now available from the spiral manufacturer) which is a useful tool in studying iron losses in the tailings stream. On our particular martite ore we found a considerable accumulation of fine-sized iron ore particles in the outside portion of the tailings stream. This fraction may be amenable to further treatment.

* Jones and Laughlin Steel Corp., Negaunee, Mich.

E. H. ROSE*—The authors have concluded an interesting piece of work and this paper is an excellent factual account of a rapid and somewhat unusual transition in practice in the plant described. Perhaps it was modesty on their part which caused them so casually to limit to one sentence the fact that "several other methods of concentration on the fine ores representative of the Hill Trumbull group . . . failed to produce consistently an acceptable grade and recovery of finished product." The fact is that they were confronted by a difficult mineral-dressing problem and they are to be congratulated on their courage and persistence in staying with it until a satisfactory solution was evolved. A visitor to the plant might have been mildly astonished, as I was, to see one type of concentrator handling that part of the load early in the 1947 season, its experimental replacement by another a little later, and then in 1948 to see that both types had simply disappeared and their place taken by the spirals which were operating as placidly as though they had been there all the time.

In working out economic methods of beneficiating Alabama red ore, most of which development is still ahead of us, it is likely that we also will pass through a period of successive disappointments, for we too have a problem where it is next to impossible either to "guess 'em right in the first place," or, because full-scale operating cost is such an important factor, to lay the ultimate answer on the line in advance by means of laboratory experimentation.

Iron ore being what it is instead of what it used to be, it is encouraging to have the example the authors have given us today that a tough nut that will not crack on the first blow or the second is apt to do so on the third or fourth.

* Tennessee Coal, Iron and Railroad Co., Birmingham, Ala.

Jaw Crusher Capacities (Blake Type)

By D. H. GIESKIENG, Member AIME

DISCUSSION

E. H. BRONSON*—I find this paper very interesting, except that I am not able to understand the derivation of the so-called "realization factor."

This factor is defined as the ratio of the size of square opening passing all the feed to the gape of the receiving opening. By this I presume the author means the size of rock feed to the width of crusher opening. So if we fed a 12 in. rock to a 16 by

* Consulting Metallurgical Engineer, Toronto, Ont., Canada.

24 in. crusher, the ratio would be 16/12.

However, this does not seem to be the case. On page 245, under the heading of Crusher No. 1, he has a theoretical factor of 1.00, with a feed of -24 in. and a jaw opening 36 in. wide.

Another point, on the same page he converts capacity of a 60 by 48 in. crusher, to a 10 by 7 in., treating hematite. With the large crusher the weight of material fed is 155 lb per cu ft, but with the smaller crusher the weight shrinks to 100 lb per cu ft.

A clear explanation of these two points would assist in reading an otherwise interesting paper.

D. H. GIESKIENG (author's reply) —In regard to the derivation of the "realization" factor, *r* outlined in the paper, the equation exclusive of *r* represents the maximum choke feed capacity of a crusher at given values of speed, setting, width, throw, and so on. The realization factor, *r* introduces the limitations imposed by hang-ups of the feed characteristic of the various practical means of feeding the crusher.

One of the probabilities of these hang-ups concerns the relative size of the larger pieces in the feed and the size of the crusher receiving opening. As a criterion of this the ratio of the size of square opening passing all of the feed to the gape of the receiving opening was employed. If a minus 12 in. rock is fed to a 16 by 24 in. crusher, the ratio would be $^{12}/_{16}$ or 0.75. From Fig 1, an average performance or "realization" of about 90 pct would be expected for well designed feeding systems, and about 50 pct for the other extreme; *r* would be either 0.90 or 0.50, respectively.

In reducing realization factors from field data it is necessary to calculate the maximum capacity of the crusher by excluding *r* and then compare this with the tonnage actually obtained. The curves shown on Fig 1 were calculated from available data.

On page 245, under the heading of Crusher No. 1, the factor 1.00 refers to the nip-angle which is 26°.

Fig 5, 6, and 7 have been used for two purposes. The first was to reduce the 10 by 7 in. laboratory crusher data to terms of the effect of setting and feed factors upon capacity. In doing this it was necessary to adjust these data to common conditions of feed density (100 lb per cu ft), stroke (0.65 in.), and speed (250 rpm). Secondly, these curves were used as a convenient means of comparison of data from larger crushers in the field. To draw this comparison as in Fig 5, the results obtained with the 60 by 48 in. crusher had to be revised to what capacity would have been obtained had the feed weighed 100 lb per cu ft instead of 155, the speed been 250 rpm instead of 163, and so on. Also, since the comparison was made at an equivalent 10 by 7 in. setting of 3 in. it was necessary to convert the nip-angle effect from 25° to 18°. Based upon 3 pct per degree this factor becomes 1.21.

It is hoped that the foregoing will help to interpret the paper. Your interest is appreciated.

F—Coal

Sampling of Coal for Float-and-sink Tests

By A. L. BAILEY and B. A. LANDRY, Member AIME

DISCUSSION

W. W. ANDERSON and G. E. KELLER*—We want to compliment the authors on this very thorough paper. It gives information which the coal industry has needed for some time. We hope that the additional information which the authors are collecting will be available shortly.

The mixing and riffling procedure that was followed for experimental purposes is obviously not practical in routine float-and-sink testing because of the particle size degradation which would result in handling the sample so many times. It is important to obtain our float-and-sink fractions with a minimum amount of handling of material.

A statement is made in the paper (p. 80) that "the variable most likely to affect the size of sample required to meet a given preassigned accuracy would be the state or degree of mixing of the coal." We agree that this is a large factor, but do not believe it is the most important factor. Our own opinion is that the most important single factor governing the total gross weight of sample that must be collected is the percentage of the weight of material in the smallest fraction that results from the screening and float-and-sink operations. In other words, size of sample is governed by the total number of fractionations that must be made, and the distribution of material within the fractions.

We can imagine a coal with perfect mixing, but with such a small amount of material in some float-and-sink fraction in one of the coarse sizes that a much larger sample would have to be taken than would be the case with very poorly mixed material, but with a large percentage of coarse material more evenly distributed in all float-and-sink fractions.

Our own observation of many float-and-sink tests that we have run in our own organization on many types of coal is that the size of sample that must be used on fine size float and sink is governed more by the requirements for weight of material to be used for analysis in the laboratory than by weight of material necessary to obtain accurate float and sink percentage of weight values. In other words, it is our opinion that very small samples can be used for float-and-sink fractionation in the fine sizes, but that accurate analysis of the fractions will depend on a larger weight of sample being pulverized for the laboratory than is necessary to establish the float-and-sink distribution with respect to weight.

* Commercial Testing and Engineering Co., Charlestown, West Va.

A. L. BAILEY and B. A. LANDRY (authors' reply)—The authors thank Messrs. Anderson and Keller for their comments based on long experience. It is agreed that the involved mixing and riffling technique used may be disadvantageous from the standpoint of degradation. Fortunately, the paper does point out that the extended riffling was unrewarding in causing further mixing. Two large unknowns remain, however: (1) how much of the mixing from the presumed highly unmixed state in the bed was achieved toward the random state during blasting, loading, transportation, screening, and further transportation to the point where the gross sample was taken, and (2) how much of the mixing took place during the preparation described preceding riffling. As has been pointed out by one of the authors,[5] the degree of mixing has a very large effect on the size of sample required and there are still too few experimental data to show at what stage of coal handling most of the mixing occurs.

The discussion states that the weight of material in a screened fraction, or in a float-and-sink fraction, is more important than the mixing factor. We do not believe that these factors are comparable in this instance inasmuch as our purpose was to give minimum sampling requirements to achieve a preassigned accuracy in the percentages of float, middlings, or sink, and nothing more. The gross sample had already been screened and no further division by screening was made or contemplated; also, it was not intended that the middlings and sink fractions would necessarily be adequate for percentage ash or other determination. In other words, the sample obtained by the method outlined is not intended for washability studies but only for preparation plant control. Further experimental work has been done, since the paper was prepared, to investigate the effect of increasingly larger top and bottom sizes on the variability of float, etc., of a double-screened coal from Western Pennsylvania. Results will be published and eventually attention is to be given to the preparation of sampling specifications.

[5] References 5 to 10 are at the end of this discussion.

E. H. M. BADGER*—I should like the authors to explain more fully the fundamental assumptions on which their Eq 4 is based. The equation is of the form

$$\sigma^2 = p(1 - p)$$

which is the usual expression for the (standard deviation)² when the chance of finding a particular kind of particle in the sample is proportional to the number fraction, p. But instead of the number fraction, the authors have used the weight fraction, $\frac{W_F}{W}$. The chance of finding a particular kind of particle in the sample can only be proportional to the weight fraction, if the average *weights* of all kinds of particles, that is, float, midlings, or sink, are the same. Surely a much more justifiable assumption would be that the average *volumes* of the particles are the same, and, if this is so, Eq 4 would not be true.

This may be demonstrated as follows:

Let w be the weight fraction of float, middlings, or sink, d_1 the density of this fraction, and d_2 the density of the rest of the coal. Then assuming that the average volumes of the pieces in the three classes are the same, the number fraction, p, is given by

$$p = \frac{\frac{w}{d_1}}{\frac{1-w}{d_2} + \frac{w}{d_1}} = \frac{wd_2}{d_1 + w(d_2 - d_1)} \qquad [5]$$

The weight fraction, w, in terms of p is given by

$$w = \frac{pd_1}{(1-p)d_2 + pd_1} = \frac{pd_1}{d_2 + p(d_1 - d_2)} \qquad [6]$$

* North Thames Gas Board, London, England.

Suppose that all the pieces of the float fraction are thought of as balls of one color and weight, while those of the middlings are of another color and weight, and those of the sink are of a third color and weight, but that the size of all the balls is the same. Then on taking samples out of a heap, the *number* of balls of one color can be noted and expressed as a fraction of the total. The (standard deviation)^2 of the number fraction will be given by

$$\sigma_p^2 = p(1 - p)$$

or on a percentage basis

$$\sigma_p^2 = p(1 - p) \times 10{,}000$$

But if the *weights* of balls of one color are noted and expressed as a fraction of the total weight, the (standard deviation)^2 of the weight fraction will not be the same as that of the number fraction, because the addition of one ball of the sink to the sample makes more difference to the weight fraction of the sink than the addition of one ball of the float makes to the weight fraction of the float.

The relation between the two standard deviations, i.e., that on a weight basis, σ_w and that on a number basis, σ_p is given by

$$\sigma_w = \frac{dw}{dp}\sigma_p$$

dw/dp can be obtained by differentiating Eq 6

$$\frac{dw}{dp} = \frac{[d_2 + p(d_1 - d_2)]d_1 - pd_1(d_1 - d_2)}{[d_2 + p(d_1 - d_2)]^2} = \frac{d_1d_2}{[a_2 + p(d_1 - d_2)]^2}$$

This can be written in terms of w by substituting for p from Eq 5

$$\frac{dw}{dp} = \frac{[d_1 + w(d_2 - d_1)]^2}{d_1d_2}$$

From the relations between p and w, it can be shown that

$$p(1 - p) = \frac{dp}{dw} w(1 - w)$$

Therefore we can write

$$\sigma_w^2 = \left(\frac{dw}{dp}\right)^2 \cdot p(1 - p) = \frac{dw}{dp} \cdot w(1 - w)$$

If the second form of the equation is used, it is not necessary to calculate p in order to find σ_w^2. The equation represents the (standard deviation)^2 of the weight fraction. If the (standard deviation)^2 of the percentage by weight is required, as in the paper by Bailey and Landry, the value must be multiplied by 10,000.

The application of these results to the Western Pennsylvania coal may be now considered. It is necessary to assume values for the mean densities of each fraction. The densities of the rest of the coal were obtained by calculation on the basis of these assumptions:

	Float	Middlings	Sink
Density of fraction	1.30	1.50	2.20
Density of rest of coal	1.75	1.365	1.325
Weight fraction	0.7705	0.1257	0.1038
dw/dp	1.192	1.074	1.526
σ_w^2 (percentage basis)	2,108	1,180	1,420

The values of the (standard deviation)^2 are substantially greater than those given by the authors, the biggest difference being for that of the sink fraction. The points should presumably be plotted against the average weight of each respective fraction and not against the average weight of the whole coal. A point that arises here is the way in which the average should be calculated. Would the authors explain why they use a simple average for float-and-sink tests but use a weighted average weight for ash?

A. L. BAILEY and B. A. LANDRY—The authors are grateful for Mr. Badger's comments on the two important points raised in his discussion.

Eq 2 of the paper is applicable strictly only to a coal whose pieces are of the same weight and, similarly, Eq 4 is applicable only when the pieces are considered to be of the same weight in which case it is of the same form as the binomial $\sigma^2 = p(1 - p)$, given by Mr. Badger, since cumulating on the weight is then equivalent to cumulating on the number of pieces.

Mr. Badger correctly points out that because pieces of sink material may have a density as high as 2.2 whereas pieces of float may have a density as low as 1.3, corresponding pieces having the same volume may have a weight ratio of 2.2/1.3 or 1.7 with a corresponding inverse ratio of the *number* of pieces offering themselves to random choice. However, Mr. Badger could have pointed out, even more forcefully, that because the size range for the bituminous coal was 1⅛ in. by No. 4, the weight ratio of the largest piece to the smallest piece, when both are of the same density, was 91. and for the anthracite of the order of 116. The effect of these ratios on choice, and on the number of pieces per increments of a given weight, far outweigh the small effect of differences in density.

These departures from theoretical requirements were recognized, but it was nevertheless assumed that Eq 2 and 4 might apply because of the compensating effects of the smaller and larger pieces when the variability of pieces or of increments is referred to their average weight. The general concordance of the results shown in Fig 3 and 4 indicate that this assumption was warranted.

Accurate calculation of the variance associated with the curve of Fig 2 would not be practicable if consideration were given to the size range as well as to the density range of pieces. On the other hand, if sampling specifications are ever prepared to cover this phase of coal sampling, a simple method must be available of calculating the variance of the single (average weight) piece from rough knowledge of the percentages of float, middlings, and sink in the coal to be sampled. This means using Eq 4 as given.

In this connection, it is hoped, of course, that from the general knowledge of the size consist of coals mined from a given bed by a specified method, it will be possible to compute the average weight of piece against which the calculated variance will be plotted. On the assumption that the size consist is the same for float-and-sink material, the average weights used may be related in the ratio of their mean densities before plotting. However, the resulting small displacements on log coordinates would probably be well within the errors of the broader assumption made on size consist and, again, we preferred to use the simpler method of not correcting for density in view of the eventual application to coals of uncertain size consist.

The second important point raised by Mr. Badger has to do with the reason for use of the simple average weight in this paper, whereas, in published work on coal sampling for ash, one of the authors has used the weight-weighted average weight.[2]

Eq 2 when expressed as Eq 1 is applicable to a coal whose pieces are not of uniform weight. The average weight of piece to be used in this instance is the weight-weighted average weight. This can be calculated if the size consist of the coal is known. For a small range in size, the difference between the weight-weighted average and the common average is small; if the size range is wide, however, use of the weight-weighted average is almost mandatory. In this paper, the common average was used primarily because no data were available on the size consist and, moreover, the size range was considered narrow enough to justify using the simpler expression, again having in mind the eventual preparation of general sampling specifications.

J. VISMAN*—The following notes refer to the basis of the theory dealt with in the paper. *Page* 79, *col.* 3, *line* 9: "At present there are no published standards for float and sink test sampling." Three years ago the British Standards Institution published a specification for screen analysis of coal (other than pulverized coal) for performance and efficiency tests on industrial plant.[6]

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Page 81, *col.* 2, "*Particle count*" *etc.*: The calculation of the weighted mean particle (the authors report to have measured the average weight of particle), would be simplified when using the sieve analysis and applying the formula

$$V = \frac{\Sigma qi \cdot d^3}{\Sigma qi} \qquad [7]$$

where V = weighted mean particle volume.

q = weight of sieve fraction, percent (so $\Sigma qi = 100$).

d = average diameter of sieve fraction.

This formula is directly related to

$$\overline{w} = \frac{\Sigma w_i^2}{\Sigma w_i} \qquad [8]$$

which may be shown as follows.

Multiply Eq 7 by the average specific gravity (γ), then substitute

$$q_i = \frac{100 w_i n_i}{W} \quad \text{and} \quad d_i^3\gamma = w_i$$

From this it follows

$$V \cdot \gamma = \overline{w} = \frac{\Sigma n_i w_1^2}{\Sigma n_i w_i} = \frac{\Sigma w_i^2}{\Sigma w_i}$$

From experiments (carried out at the Dutch State Mines) it has been found that coal particles of a sieve fraction $(d_1 - d_2)\Phi$ may under laboratory conditions, be considered within narrow limits as cubes with sides $d_{12} = \frac{(d_2 - d_1)\Phi}{2}$.

In case round sieve-holes are used, the particles of a sieve fraction $(d_1 - d_2)\Phi$ may be considered as cubes with sides

$$d_{12} = \tfrac{1}{2}\sqrt{2}\,\frac{(d_2 - d_1)\Phi}{2} = 0.7\,\frac{(d_2 - d_1)}{2}\,\Phi$$

according to the ratio between the side of a square and the diameter of the circumscribed circle.

The ratios given here are valid under laboratory conditions only.

The weighted mean particle volume of a coal may be computed accurately and quickly from the size-consist data, provided the coarse fractions are kept within narrow limits.

Thus the weighted mean particle weight ($\overline{w}$) too may be determined directly from the size-consist data, without the use of a planimeter, the Rosin-Rammler equation, or even the counting of particles.

Page 82, *col.* 2, "*Calculation of* $\sigma_w{}^2$": In the formula for the variance ($\sigma_w{}^2$) the influence of the specific gravity has not been mentioned.[6] As the accuracy of a specific gravity separation is as a rule expressed by the accuracy of the smallest fraction (by weight), one might make use of the following formula,*

$$\sigma_x{}^2 = 100\,\frac{x \cdot \gamma \cdot V}{W}$$

σ_x = standard deviation of (x) in percentage of (W).

x = weight of smallest fraction in percentage of (W).

γ = specific gravity of fraction (x).

V = weighted mean particle volume in (cu cm Φ).

W = weight of sample, grams.

which is valid where $x \leqq 10$ pct, and on the understanding that there is no segregation (to be used only when reducing a gross sample under laboratory conditions). In case of sampling segregated coals, there should be added a variance ($\sigma_s{}^2$), which is proportional to the rate of segregation (B) of the coal, and to the number of increments (N) as follows:

$$\sigma_s{}^2 = \frac{B}{N}$$

As the influence of the analysis on the accuracy is negligible as far as specifications are being strictly adhered to, the ultimate variance (σ^2) may be formulated as follows

$$\sigma^2 = \sigma_x{}^2 + \sigma_s{}^2$$

$$\sigma^2 = 100\,\frac{x \cdot \gamma \cdot V}{W} + \frac{B}{N}$$

where W = weight of gross sample in grams.

B = variance of (x) due to segregation.

N = number of increments (for $x \cdot \gamma \cdot V$ see above).

This formula is valid if the gross sample is analyzed as a whole, without reducing its weight. If on the other hand the gross sample (W) would be reduced in weight to (W_1) grams before analyzing, another variance ($\sigma_r{}^2$) would have to be added

$$\sigma_r{}^2 = 100\,\frac{x \cdot \gamma \cdot V}{W_1}$$

So, $\sigma^2 = \sigma_x{}^2 + \sigma_s{}^2 + \sigma_r{}^2 + \ldots$

If the process of splitting would be repeated, again a variance $(\sigma_r')^2$ would have to be added, etc.

The variances $\sigma_r{}^2$, $(\sigma_r')^2$, $(\sigma_r'')^2 \ldots$ form a convergent geometric series, the sum of which tends quickly to a limit with increasing number of subdivisions

$$\bar{\sigma}_r{}^2 = 200\,\frac{x \cdot \gamma \cdot V}{W_{\min.}}$$

Here ($W_{\min.}$) is the final weight of the sample intended for analysis. So the total standard deviation follows from:

$$\sigma^2 = 100\,\frac{x \cdot \gamma \cdot V}{W} + \frac{B}{N} + 200\,\frac{x \cdot \gamma \cdot V}{W_{\min.}} = \frac{A}{W} + \frac{B}{N} + C \qquad [9]$$

Here A, B, and C are constants for a given product, at a given method of sample reduction.

Page 84, *col.* 3 *line* 5: "The general agreement . . . gives confirmation to Eq. 2 and justifies its use in establishing sampling characteristics of coals for determination of float and sink values; and, incidentally gives added confirmation to the use of the similar Eq 1 in coal sampling for average ash."

From the mathematical point of view there is some doubt about the validity of the "mixing exponent."

If the above-mentioned Eq 9 is expressed in the form of Eq 2 of the authors, we find:

$$\sigma_W{}^2 - \frac{B}{N} = \left\{100x + \frac{200W \cdot x}{W_{\min.}}\right\}\left(\frac{W}{\overline{w}}\right)^{-1}$$

Plotted on a log-log diagram there ensues a straight line slanting under 45° from the point ($\overline{w}$, $\sigma_{\overline{w}}{}^2$) as long as the influence of the segregation $\left(\frac{B}{N}\right)$ and of the subdivision of the total sample are negligible as compared with ($\sigma_W{}^2$). In other words, from the mathematical point of view a curve may be expected which starts from the point ($\overline{w}$, $\sigma_{\overline{w}}{}^2$) as a straight line under 45°, and which will be bent up slightly at the lower end, as a result of the influence of the segregation and of the reduction on the total variance ($\sigma_{\overline{w}}{}^2$).

The same conclusion is valid for ash determinations. Here the formula for (σ_W) is[9]

$$\sigma_W{}^2 = \frac{A}{W} + \frac{B}{N} + C$$

A/W = variance due to size consist and ash distribution.

B/N = variance due to segregation.

C = variance due to reduction and analysis.

Putting this formula again in the form of Eq 1 of the authors, we find:

$$\sigma_W{}^2 - \frac{B}{N} - C = \frac{A}{\overline{w}}\left(\frac{W}{\overline{w}}\right)^{-1}$$

From this it follows theoretically, that, as long as $\left(\frac{B}{N}\right)$ and (C) are small in comparison with ($\sigma_W{}^2$), the curve on the log-log diagram will start rectilinear at point ($\overline{w}$, $\sigma_{\overline{w}}{}^2$), sloping at 45°, bending upward at the lower part. In our opinion therefore, the extrapolation by means of straight lines does not correspond with the theory of probabilities.

It is expected that if the data mentioned in the publication under consideration are being reconsidered from this point of view, correcting the weighted average weight of particle and its variance ($\sigma_{\overline{w}}{}^2$), a similar picture will be obtained.

From experiments carried out at the Dutch State Mines it was found that the

* Mentioned earlier in a different form by Kassel and Guy;[7] for derivation see ref. 8

factor $\frac{A}{\bar{w}} = \frac{A}{V\gamma} = \sigma_{\bar{w}}^2$ ranges (for the ash determination) from 0 — ca 2000, a value much higher than the variances found by Landry,[10] while (B) ranges from 0 — ca 200.

$\left(\frac{A}{V\gamma}\right)$ is closely related to the ash-content, whereas the variance (B) depends on the form of the washability-curve as well as on the method of cleaning (compare products of jigs and heavy-medium washers).

Evidently, the variance ($\sigma_{\bar{z}}^2$) can never be measured directly from a multiple-size product, because the smallest sample which may be considered as representative for that product should have a weight equal to at least several times the weight of the coarsest particle, that is (for a product corresponding to the formula of Rosin-Rammler) about 10 (as an order of magnitude) times the weight ($\bar{w}$); so the variance (σ^2) only has real meaning for values less than about 200.

There remains much to be said on this subject, particularly with respect to the influence of small samples on the accuracy and the question of bias.

In our opinion it is possible to draw up specifications for the sampling of coal and ore (for the analysis of several constituents, size consist and specific gravity separation), based on the principles mentioned above. The ultimate form in which these specifications are given, may be very simple. In the case of ash determination for instance, it is possible to express the weight of sample and the number of increments directly as a function of the total ash-content and of the maximum particle size, at a given accuracy.

The experimental work needed for these specifications may be restricted to a minimum. The main difference between these specifications and those now in use is that the number of increments would be considerably higher, whereas the minimum weight of increment would be diminished—in extreme cases—to several times the weight of the coarsest particle.

A. L. BAILEY and B. A. LANDRY—The authors are in agreement that the extended remarks of Dr. Visman are directed toward the published works on coal sampling of the junior author of the paper rather than at the paper itself.

Dr. Visman's Eq 7 for the weighted mean particle volume would appear to have considerable merit except that it substitutes a step-wise process for the continuous process which results from use of the Rosin-Rammler equation or the planimeter method; thus, a somewhat larger number of screen sizes should be required for the same accuracy. In planned experiments this can be achieved but in using available information from commercial operations fewer than enough screen-size data can be expected as a rule.

Eq 9 is well known to the junior author as it has been suggested or used by a number of statisticians concerned with sampling. Data have not been presented in the literature, however, to show that it will apply; that is, that A and B will remain constant, when the variance of the percentage ash of coal is determined experimentally for increments of increasingly larger weights, with the understanding that these are to be reduced to laboratory size by a process that involves crushing of the entire increment before riffling or quartering. Upon such data depends ultimately the choice of formula for the variance.

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Synthetic Liquid Fuels from Coal

By J. D. DOHERTY, Member AIME

DISCUSSION

A. R. POWELL*—Mr. Doherty has outlined in a most thorough manner valid arguments for the development of an industry in this country making synthetic liquid fuels from coal. No thoughtful person will dispute the statement that it is essential to carry on fundamental and applied research and acquire engineering "know-how" by the erection and operation of two or three large plants in the near future. Considerations of national security indicate that this should be done, irrespective of any temporary present surplus of natural petroleum.

It is the belief of the discusser and of many others interested in coal technology that our energy picture of the future will include an increasing utilization of coal directly, without any conversion to liquid fuel, as a source of energy. A fact that is sometimes forgotten is that petroleum fuels have often been cheaper than coal in many locations and over long periods of time, and this price factor has been of great importance in expanding the use of petroleum fuels at the expense of coal. Synthetic liquid fuel made from coal must necessarily be several times as costly as the coal from which it is made, and this fact alone will have a profound effect on the future energy pattern of our country. This complete change in the economic relationship between solid and liquid fuels will lead to the substitution of coal itself for many uses now supplied by the relatively cheap petroleum fuels.

From the standpoint of conservation of our natural resources, the substitution of coal itself for liquid fuels wherever possible when the age of synthetic liquid fuel arrives is also logical. At least 50 pct of the energy of coal is lost during conversion to liquid fuel. Nor is this compensated for by a higher utilization efficiency of the liquid fuel in most cases. For example, modern steam-boiler plants can operate at equal fuel efficiencies with either coal or fuel oil.

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It seems to the discusser that Mr. Doherty has underestimated the curtailment of oil consumption that will accompany the future introduction of synthetic oil made from coal. Reduced oil supplies need not curb our economic advancement, as Mr. Doherty fears, provided proper research and development programs on improved utilization of coal itself as an energy source are carried out. Such programs should be given as much emphasis as the present synthetic-oil programs in the interest of conservation, national security, and economy in our fuel planning for the future.

Unless some revolutionary development now unforeseen occurs, liquid fuels will always be predominant for use in the various forms of internal-combustion engines and for certain specialized fuel uses, even though their cost is several times that of coal. For this reason it is important that much research and development on synthetic oil from coal be carried out ahead of the time that decreasing petroleum production and higher prices make the manufacture of synthetic oil necessary, and the Bureau of Mines is to be congratulated on its foresight in aggressively following such a program. Despite the belief expressed above that direct substitution of coal will cause curtailment in liquid-fuel use when the synthetic era arrives, the discusser also thinks that the synthesis of liquid fuel from coal will prove to be one of the major developments in this country during the next decade or two.

An Evaluation of the Performance of Thirty-three Residential Stoker Coals

By JAMES B. PURDY and HARLAN W. NELSON

DISCUSSION

C. F. HARDY*—When a new mine is opened, there is always a question as to the suitability of the coal for various uses including domestic stokers. Until this service was offered by Battelle, it was customary to hand-screen a few hundred pounds of coal and distribute it among various engineers or other stoker users. Obviously, this is unsatisfactory and in no sense of the word standard procedure. There have been several instances where stoker plants were installed by this hit-or-miss system of evaluation, and the coal has proved unsatisfactory for use in domestic stokers after the plant was built. On the other hand, the Battelle standard evaluation tests, by showing that the coal was unsuitable or borderline, have prevented several companies from useless investment in stoker-screening facilities, and the cost of trying to promote the coal on the market.

This paper is particularly valuable in that it brings to the attention of the coal industry the fact that there is a method of testing available, which gives a reasonably accurate evaluation of a coal for domestic-stoker use before it is placed on the market. This should stimulate further research which should tell why coals perform as they do in domestic stokers.

R. J. HELFINSTINE†—The correlation shown between the laboratory combustion tests and the public acceptance of the coal is of particular interest. This type of information is needed to bridge the gap between the laboratory and the consumer.

This paper furnishes more evidence that "armchair philosophy" is often used to condemn good stoker coals. Obviously, no coal should be branded as unsuitable for domestic stoker coal because of high (or low) ash-fusion temperature, or high free-swelling index. Actual combustion tests are required to establish the clinkering and coking characteristics of a coal, and judgment should be based upon performance and not upon appearance.

The tests described in this paper show that a longer hold-fire period was required with high ash coals. This might be expected with the type of hold-fire control used for the tests. However, it is the writer's opinion that this relationship would not exist if the controls were of the time-interval type.

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† Illinois State Geological Survey, Urbana, Ill.

C. H. SAWYER*—The work described in this paper represents the longest-standing program of such research with which we are familiar, and all of us who are interested in this subject have borrowed freely both in ideas and in method from it.

Both R. I. Bush, Director of our Pittsburgh Stoker Research Laboratory, and the writer had an opportunity to examine this paper in advance and he shares with the writer responsibility for the following comment:

We liked especially the authors' careful statement that values of fuel-bed resistance indicate ". . . the general condition of the fuel bed." Certainly, fuel-bed resistance cannot be ascribed to any one property of the fuel bed, such as thickness, as is so often claimed. We think this error has led to many false starts on air-control design and it certainly finds its way into much stoker advertising. We think the limit of 1.5 in. of water given as maximum trouble-free fuel-bed resistance exceeds the capabilities of several stoker fans we have tested and would be safer set at some lower value, say, of the order of 1.25 in. Fortunately, there is a trend toward higher-pressure fans among stoker manufacturers, and the point will assume less weight as time eliminates the old low-pressure jobs.

The authors have questioned the temperature selected for hold-fire pickups in their method. We use an entirely different method of evaluating hold-fire performance. In typical stoker operation out fires are most likely to occur at some time between 3 and 5 hr after the last prolonged operation of the stoker. Many coals, particularly the semifluid types represented by some of the popular low-volatile stoker coals, will pick up very sharply after this low-activity period and would thus show an average time of hold-fire operation little if any below the normal, even though their remaining alive

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may have been a touch-and-go proposition at the critical stage. We cannot, however, offer a completely satisfactory substitute for the method cited. We still use 3 min per hr timed operation as a hold-fire test. Most coals will hold fire at such a rate, and we think minimum activities, as indicated by stack-temperature records, give a closer indication of the likelihood of out fire than does the average reactivation time of the Battelle method. However, we pay for this policy with occasional out fires which means virtual inability to run a "standard" test on the coal in question since much other data of value are no longer reliable after rekindling such a fire. Possibly avoiding a hold-fire check until the end of a test schedule might overcome the objection to some such attempt to get a closer assay of hold-fire characteristics.

Mr. Bush has suggested that residual coke as determined is subject to too much accidental variation to allow significance by any means short of running several tests on each coal to obtain an average value. He does not feel that the size consist of residual coke is subject to as much variation as is the quantity, and suggests comparison of some figure such as average size of residual coke as an alternative attempt to index the likelihood of coke-tree trouble.

We find the lack of pattern in clinker data disappointing as did the authors. It seems that no test of such short duration can answer satisfactorily the question of "clinkerability." However, in our own laboratory we have noted a slightly better correlation of clinker removed with ash-softening temperature than that indicated in the graph shown. The writer has wondered considerably about this difference since he knows how carefully this work is done at Battelle. One possible partial explanation for the difference is offered:

Our fires are undisturbed throughout the entire week. In the tests under discussion clinker is removed on the evening of the fourth day. It would seem only human nature for an operator to search a fuel bed showing little or no clinker more thoroughly than one containing some obvious pieces at this stage of the test. Such search would constitute cultivation which is always of help in clinkering difficult ash. Thus certain high-fusion or otherwise difficult-to-clinker coals might be unconsciously thrown out of line in this performance characteristic by even the most conscientious operator. As a matter of fact even an equivalent degree of cultivation (unavoidably entailed in any clinker search) might favor high-fusion coals in this regard and tend to even the end quantities of clinker.

J. B. PURDY and H. W. NELSON (authors' reply)—Mr. Hardy has pointed out the economic advantages of a laboratory conducted evaluation test. This of course is of prime importance. In addition, one of the strong points in favor of a test method of this type is that it permits the evaluation to be made regardless of the season or of fluctuations in outside temperature. It also permits making a relative evaluation of coals with comparative ease, since consideration of variations in equipment is reduced to a minimum.

Mr. Helfinstine's comment regarding the type of hold-fire control used is correct. However, as pointed out by the authors in the paper, the method used does provide a relative means of comparing the performance of coals during the periods of hold fire, even though it must be recognized that the period of operation during each hour may be longer than necessary. The method eliminates interruptions occasioned by attempts to find the lowest possible hold-fire setting for each coal under test, yet it furnishes a relative measure of performance.

Mr. Sawyer's discussion of clinkering data is of special interest. Disturbing the fuel bed certainly has an effect on the subsequent formation of clinker in spite of all the care that is taken in handling the clinker tongs. However, when burning a high-ash coal at a feed rate of 30 lb per hr, as employed in these tests, sufficient ash has been deposited by the end of the fourth day to make it often necessary to remove the clinker at that time, thus insuring a relatively clean fuel bed for the intermittent heat-demand run of the last day of the test.

Some Aspects of Mechanical Coal Cleaning in Utah

By CARL S. WESTERBERG

DISCUSSION

(*H. F. Yancey and Orville R. Lyons, presiding*)

L. C. McCABE*—An increased demand for coal in the west is to be expected because of the growth in population and industry during the past ten years. The author calls attention to the increased mechanical cleaning of Utah coal between 1938, when no cleaning was being done, and 1947, when more than 24 pct of the production was cleaned. This is essentially the period of most rapid growth of the Pacific Coast area although the growth and, no doubt, the general trends in coal production and cleaning continued at a less accelerated rate during 1948. The Pacific southwest increased 27 pct in population between 1940 and 1947 while the United States showed only a 7 pct increase in the same period. It has been estimated that there will be an additional 15 pct increase in population between 1947 and 1950. The conclusion that more coal and more mechanical cleaning capacity will be required appears to be well founded.

Mr. Westerberg discusses the economic advantages of coal cleaning reflected in freight savings, in lower cost per million Btu, and lowered ash and sulphur content. In view of the limited occurrence of coking coals in the west it would be interesting to know whether significant beneficiation in coking strength can be expected from mechanical cleaning of Utah coals. This has been demonstrated for some eastern and midwestern coals.

One can readily agree to the desirability of recovering the fine coal that has heretofore been lost in washery slurry. The very cheapness of coal has hindered

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the development of fine coal recovery in the past but with improved fine coal recovery methods it is possible that recoveries of washed coal will approach the theoretical values of the washability curves.

In European preparation practice, fine coal is more widely recovered than in the United States. There are a number of reasons for this, some of which are that coal is generally more friable and fine coal recovery is essential, it is mined with greater difficulty and greater cost and commands a greater share in the price of finished products. Europe is geared to a coal economy and in the past oil has not been important for power generation and heating purposes. This is another way of saying that our very abundance of both coal and oil is responsible for this difference in recovery practice.

Coal Washing in Colorado and New Mexico

By J. D. PRICE and W. M. BERTHOLF, Members AIME

DISCUSSION

(*H. F. Yancey and Orville R. Lyons, presiding*)

A. C. RICHARDSON*—First of all, I think that the paper represents a lot more work, study, and correlation than has been indicated by the brief talk by Mr. Price. I like the way he started out and described the areas from which the samples were obtained, the locations of the washing plants, the available tonnages, and other background information with which to evaluate the data he submitted later on. Then I like the way in which he described the various types of washing plants, the tonnages handled and the difficulties of the washing problems; showing the amount of material that lies close to the specific gravity at which the washing separation is made. Later he gave figures from washing plant operations showing recoveries and cleaning efficiencies.

He then discussed his own plant at Pueblo. It is the same old plant, I think, that I worked around a good many years ago. It is unusual to find a plant treating nearly 5000 tons of coal a day on tables. But this table plant is, I believe, more efficient than is indicated by the figures that Mr. Price gave.

To determine the efficiency of a cleaning operation or to compare it with another it is necessary to consider the quantity and character of the material close to the specific gravity at which the separation is made. It is not fair, I believe, to penalize the table operation by something like 4 pct of out-of-place-material as he has done here. The variety and difficulty of the coals that he has to wash, the continuous shift and change in their composition make a very difficult cleaning problem and the table performance is excellent.

I believe that the information in this paper will be of interest and value to anyone operating or planning to build a coal cleaning plant in this or other areas; particularly where the cleaning of fine coal is a problem. The data may be used for comparative purposes in determining the relative efficiencies of other cleaning plant separations.

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E. D. HAIGLER*—What is a Baum jig?

* The Foxboro Co., Kimberly, Nevada.

J. D. PRICE (authors' reply)—A Baum-type jig is one in which the pulsations of the water is secured by means of a pulsating air current applied on top of the water. I imagine you are all familiar with the old plunger-type jig which is in effect a U tube in which a plunger on one side of the U, moving up and down, causes a corresponding pulsation on the far side of the jig. In the Baum jig, the pulsating air current is applied on the surface of the water on one side of the U tube of the jig and gives a corresponding pulsation on the other. It is also commonly known as a pneumatic jig. The control of the rise and fall of the water in the jig body proper is under much better control than it is in any of the other type jigs. Mr. Richardson could enlarge on that feature, for I know that he has had considerable experience with these jigs.

A. C. RICHARDSON—You have asked how to control a Baum-type jig. The pulsations in a Baum jig can be modified and regulated to a marked degree by the amount of water admitted to the jig and by the adjustments of the valve which regulates the manner in which air is admitted.

The number of pulsations per minute is controlled by the number of cycles of the air valve. Thirty to forty cycles per minute is a good speed for large jigs treating coarse sizes of coal.

With an air valve it is possible to modify the time-velocity curve of the pulsating water to some extent which in turn determines the action in a jig bed. Within limits the following parts of the air valve cycle may be regulated: (1) the rate and period of air admission, (2) the period of air expansion, (3) the rate and period of air exhaust, and (4) the period of air compression.

The rate and period of air admission determines the acceleration of the water at the beginning of the pulsion stroke and the amplitude of the stroke. The period of air expansion, after inlet port is closed, is one in which the water has reached the desired velocity, positive acceleration reduced, and the bed held in a mobile condition. The rate and period of the air exhaust can be adjusted to modify the degree of suction and so modify the manner in which the particles in the bed stratify. The compression period, after the exhaust port closes and before the intake port opens may be used to advantage in retarding the downward velocity of water during the suction stroke.

An ideal jig stroke is one in which during the up stroke the bed is lifted slowly in a mass and opens up like an accordian with the bottom layers dropping away first. With the bed open and mobile the particles adjust themselves according to their hindered settling rates. During the down stroke, while the bed is still open the particles of high specific gravity are accelerated toward the bottom layers. It is possible to approach this stroke with all types of jigs but it is less difficult to approximate it with a Baum jig.

C. S. BLAIR*—I would like to know what size coal is treated on these Plat-0 tables, and whether that had some bearing on whether they were less efficient than these Baum jigs. I have never heard anywhere that a table was not more efficient on fine coal than any type jig.

J. D. PRICE—The size we are now treating is ½ in. by 0 with no fines taken out ahead of the tables. The entire supply of coal is crushed to ½ in., and all sizes are treated on the tables. The reason that the tables are not as efficient now as in 1940 was shown on Table 18 which gives comparisons of washed coal for 1940 and 1947. The coal now contains such a great amount of impurities, such a great amount of "middling" gravity material, that the tables are considerably overloaded with reject material. As a result, it is impossible for the riffles to completely carry the middling material to the reject end and a considerable amount goes over the riffles into the washed coal.

The coal, coming as it does from twenty or more mines, varies considerably in quality and the tables can be adjusted only for the average quality of the coal. When the coal is of better quality than this average, good coal goes into the refuse; when it is of poorer quality than average, material which should be rejected finds its way into the clean coal. Hence the average overall efficiency of separation is not as high as might be desired.

C. S. BLAIR—You overload the tables?

J. D. PRICE—That is right. The coal is not suitable for high speed operation on the tables but we must run at a high capacity to get the required tonnage through.

* Black Diamond Coal Mining Co., Birmingham, Ala.

Coal Washing in Washington, Oregon, and Alaska

By M. R. GEER and H. F. YANCEY, Members AIME

DISCUSSION

(*H. F. Yancey and Orville R. Lyons, presiding*)

O. R. LYONS*—I know that we are all interested in hearing about problems that other people have. To most of the people from the eastern part of the United States, this kind of coal preparation is completely different. It represents washing difficulties that most operators have never experienced, or, if they have, they have refused to consider them because there are coals that are much easier to wash and, therefore, they leave the more difficult ones alone for the time being.

E. R. McMILLAN†—I think the authors have summarized the material very well. Dr. Yancey has spent a great many years in the study of washing coals, as has Mr. Geer. I might comment briefly on some of the plants that were mentioned, plants with which I have had some connection, particularly those of our own company, the Northwestern Improvement Co. I would like to point out this: that in each case, a great deal of work was done in the way of preliminary studies of washability of the coal before plants were built. The first plant of any size built in Roslyn field was mentioned in the paper. This central cleaning plant was designed and built to handle coal from three different mines. I might say that prior to 1935, the year in which the plant was built, the coal was used largely as railway fuel, close to a million tons annually. But as mining increased in depth, it became increasingly difficult to separate and gob much of the impurities underground, with the result that the run-of-mine coal became so high in ash that the railroad eventually was forced to consider installing some kind of preparation plant at the mine.

We spent about a year in preliminary studies in cooperation with the Bureau of Mines to find out the best type of plant that would clean the coal. The result, as Mr. Geer has explained to you, is a rather complicated plant, but it is doing a very satisfactory job. We have, however, made a number of changes and improvements since it was first built. The last addition was the installation of a centrifugal drier to dewater the minus ¼ in. size. We are now contemplating installing heat driers for further drying of minus ¼ in. We use Vissac heat driers for drying the stoker coal, the ¾ to ¼ in. The demand for drier coal, for both railway and industrial use, is forcing us to further drying of the coal.

The plant that was mentioned as having been built in Pierce County, during the war, was a government sponsored plant. I happen to have had some connection with its design, construction and initial operation. We did what we thought then, and still think, was a very thorough job of preliminary studies in cooperation with the Bureau of Mines on the coal from that field. The plant was as good a plant as could be built with the equipment available at that time, 1943. The plant that was mentioned as having been built at Ravensdale for treating the McKay coal was rebuilt about two years ago, and in addition to the modified Elmore jig washer that was mentioned, we added a concentrating table for recleaning the minus 3/32 in. Our object there was to produce a commercial

* Battelle Memorial Institute, Columbus, Ohio.
† Northwestern Improvement Co., Seattle, Wash.

stoker coal with an ash content of 5 pct or less.

W. M. BERTHOLF*—In the data shown on minus 3½ or minus 3 in. coal, there was one coal which was sized and data shown for various fractions from 1 in. down. It would appear that these coals are extremely streaky or laminated and would have to be crushed very fine in order to liberate the impurities. Is this the situation?

* Colorado Fuel and Iron Corp., Pueblo, Colo.

M. R. GEER (authors' reply)—That would follow for coking coal where the market would permit you to do such fine crushing; all steam or domestic coal cannot be reduced down to minus ¼. However, what you say is true; reducing the coal to that top size would give a higher yield at a lower ash content.

A Technical Study of Coal Drying

By G. A. VISSAC, Member AIME

DISCUSSION

(*Joseph Daniels and J. W. Woomer, presiding*)

O. R. LYONS*—I wish to thank Mr. Vissac for his compliment. I hope that his paper is not only well received, but that it will serve to bring forth more papers on the subject of thermal drying. One of the primary purposes of the work performed by Battelle for Bituminous Coal Research in investigating the thermal drying of coal was to stimulate other investigators and to get them to contribute their knowledge in the form of papers such as this one. We at Battelle and the personnel of Bituminous Coal Research are very gratified that Mr. Vissac and other persons have responded in this matter of the thermal drying of coal.

I wish to state that I think that Mr. Vissac's paper is a very clear and easily understood description of a method of calculating the design requirements for a screen type drier, and I think that it would be exceedingly valuable to operators and to those who intend to purchase any type of thermal drier and use it in the future, if the manufacturers or operators who have such information for other types of driers would provide the same type of information for the other makes of driers now on the market.

I also wish to point out—an idea that is new to me, and I know is new to most of the operators of driers in the United States—the idea of recovering the heat that is normally lost in the coal and in the exhaust gases. This heat is not being recovered at most of the thermal drying operations in the United States, and the possibility of recovering it should be called to the attention of every single one of those operators. I know many of them have never given any thought to the matter, but they will be interested once they realize the ease with which it could be done and the savings that could be realized.

* Battelle Memorial Institute, Columbus, Ohio.

I also wish to compliment Mr. Vissac for presenting the method of analysis that he uses to determine the difficulty of drying any particular coal. It is a very simple method, and yet it seems to me that it should be a very effective, very efficient method for determining the difficulty of drying for his particular problems.

C. P. HEINER*—I do not know that I can add anything very illuminating to what Mr. Vissac has said. I think anything that Mr. Vissac said in regard to coal drying is a contribution because, to my personal knowledge, he has studied the matter carefully for many years and made many valuable contributions.

I am not too familiar with coal drying problems in the east, but I know in the west we have not made enough coal drying studies. I think coal operators too often just take the coal as it is and make more or less the best of it. There are relatively few washing plants in the west now, and so the problem has not come to the front as much as it probably will in the future.

In this connection, it seems to me that this matter of drying the raw coal, as Mr. Vissac brings up, is an extremely important one. We have not a continuous miner ourselves, yet, but we expect to get some this year, and we think the percentage of fine coal—that is, minus $\frac{3}{16}$ in.—will double. We have about 20 pct minus $\frac{3}{16}$ in. in the 8 in. by 0 size now, and we think we will likely have 40 pct, which will have a surface moisture of the order of 8 pct. To wash it satisfactorily, we will have to dry the raw coal first in order to screen it, and after that, I suppose, there will have to be dry cleaning of some sort.

* Utah Fuel Co., Salt Lake City, Utah.

We have not really used dry cleaning on fines in the west yet to my knowledge, but it is a matter that has to be faced by the industry, and I am very hopeful that Mr. Vissac's study will assist us in that connection.

W. L. McMORRIS*—In my company we are preparing largely metallurgical coal for a great number of byproduct coke plants. The most outstanding thing to me about the requirements of moisture in the finished product is that there is a different requirement for almost every coke plant. Each operator has a different set of factors on which he establishes his coking costs where they involve moisture. For our corporation operations in Birmingham, my company does not produce the coal, but in Birmingham they are getting away with moistures very much higher than our plant at Clairton, Pa., would tolerate. The moisture that we have to produce for the plants along the lakefront where they are subject to much more severe weather is something else again. We have not tackled heat drying, primarily because our customers do not know what heat drying will do to the coking characteristics of the coal. If the temperature of drying can be held down

* H. C. Frick Coke Co., Pittsburgh, Pa.

to from 120 to 140°, it is hard to imagine that heat drying would affect the coking characteristics unless they were on the ragged edge to begin with.

The mechanical drying of coal has been given a great deal of study recently owing to the advent of the solid bowl centrifugal filter that has been installed in several plants and sold by the Bird Machine Co. of Massachusetts. Other filters of that same character, in smaller sizes, have been used widely in other industries. We have had some experience with the Bird filter and have found it desirable for the mechanical removal of moisture. They are efficient, but one of the outstanding reasons why it is, let us say, a desirable unit, is that it does give a mixture of all the sizes which are going to be put to that type of mechanical drying. In drying ¼ by 0 material the Bird filter cake contains all of the fine sizes. The handling of coal by vacuum filters and screen basket driers gives two fine-mesh products. Our experience with the Bird filter has been variable. Thirteen were recently installed in a plant but for our conditions we found it desirable to rebuild them. We had done some experimental work as to the most desirable bowl contour to secure the most efficient drying. We had formed that bowl contour with baffles making a coal bed. When formed with steel, the center section had an angle of about 19° with the rotating axis of the machine, and we found that the coal would not convey up that slope. It would hang until it was forced through. Instead of carrying a normal bed, it built up and would move by fits and starts. We put in what our men in the plant called "skid chains." We took the machines apart and put in conveying strips in the line of normal travel of the material to prevent its moving around the circumference of the bowl. We do not know yet how good a job of drying we are going to do. It has been irregular, but the whole operation of the new plant has been irregular, and we have not said yet that we cannot make a regular cake. We did have mechanical difficulties most of which are corrected.

Whether the Bird filter would put coal in the best condition for thermal drying is something we do not know. As I said before, our biggest problem is knowing what our various customers have to have in the way of a final product. I see there are some coke plant men here, and I would not be surprised if they have different ideas of what they need in their plants.

G. A. VISSAC (author's reply)—I think we agree entirely, at least that is what I tried to convey, namely: all possible moisture should be removed mechanically before heat drying. We have had some experience with mechanical driers, too. We have the centrifuge, but did not buy the Bird centrifuge because with our coal, it was not possible to reduce its moisture below 12 pct. In other words, the final moisture content that a centrifuge would be able to give varies so much with the kind of coal and the size composition, that it is very difficult to guarantee a moisture content.

When it comes to requirements, I am at the present time working on a new coke plant in connection with Simon Carves of England. They have a new design by which they claim the higher the moisture content, the better they like it. In fact, they want me to use 10 pct moisture, which is unusual as my experience has been that in coke preparation 6 pct was about the ideal. It is certain that with more moisture, even if the brick walls can carry it and if nothing is disturbed, time is required to evaporate the water. As it takes as much as 2 or 3 hr to evaporate the water, the capacity of the oven is reduced accordingly. In addition, we have not only to deal with coke, but have more to do with briquettes, which the railways are using. All the coal in the old country used by the railways is briquettes, because in this way there is a definite standard of quality, and it is easier to store.

To make a good briquette, the moisture cannot exceed more than 2 pct. That is where our drying problems become difficult. Railway specification at the present time on mine run, to show that moisture is far more important than ash, is as much as 16 pct ash, and less than 3 pct moisture. That is why we have to dry it.

The results of centrifuge driers in our country are that we are never able to get below 9 to 12 pct moisture. Coals freeze below 3 pct, so the centrifuge does not solve our problems.

Drying Low-rank Coals in the Entrained and Fluidized State

By V. F. PARRY, J. B. GOODMAN, and E. O. WAGNER

DISCUSSION

(*Joseph Daniels and J. W. Woomer, presiding*)

C. P. HEINER*—If you take out 35 pct of the total weight of the coal in the form of moisture, would that be about what it was in the case of North Dakota lignites?

V. F. PARRY (authors' reply)—That is about it.

C. P. HEINER—That will be about a 35 pct loss and the final product will be about 11,000 Btu?

V. F. PARRY—That is right. The heating value of the dried lignite is 10,000 to 11,000 Btu.

C. P. HEINER—And the thought is not, then, to produce a commercial coal with that heavy loss, but to make other products—that is, synthetic fuel base, is that it?

V. F. PARRY—The principal thought is to upgrade the coal, making it a more favorable raw material for industrial purposes, both for shipment and direct use. You see, the trouble in these lignites is that their moisture penalizes their

* Utah Fuel Co., Salt Lake City, Utah.

shipment, and if we can remove that at low cost, they can be shipped greater distances to industrial plants.

C. P. HEINER—The only observation I have to make is if you only raise it to 11,000 Btu you still have a vast quantity of reserves that are much hotter, as you well know.

V. F. PARRY—This is just the long range study of possibilities of these low grade coals, and raising them up to 11,000 Btu does not reach the quality of high grade coals yet.

The Rupp-Frantz Vibrating Filter

By W. M. BERTHOLF and J. D. PRICE, Members AIME

DISCUSSION

(*Joseph Daniels and J. W. Woomer, presiding*)

W. J. PARTON*—I have not had the opportunity to read this paper, and I do not have a written discussion. However, I thought it might be interesting for me to relate some of the experiences we had with equipment similar to the vibrating filter as described by the authors.

At the Tamaqua flotation plant of the Lehigh Navigation Coal Co. approximately 40 tons per hour of froth concentrate carrying 60 pct by weight moisture are produced. The major problem encountered at this plant is the dewatering of this coal froth so that a satisfactory product can be sent to market. In the original design of the plant a centrifuge of solid bowl type was included for dewatering this material. The centrifuge did not work out as well as we had hoped. High maintenance costs and moisture content in the cake were obtained.

A Robbins dewatering screen was installed at a later date with the idea of using it in conjunction with the centrifuge. The froth concentrate from the flotation cells was fed directly to the Robbins dewatering screen. The cake from the screen carried approximately 55 pct of the feed solids. Moisture in the cake was approximately 24 pct by weight. The underflow from the screen carried 45 pct of the feed tonnage at about 80 pct moisture by weight. The underflow product was then pumped into the centrifuge with the idea of using the centrifuge for recovering the tonnage lost through the screen. This circuit did not operate as satisfactorily as we expected. The only benefit derived was in the reduction in the power consumed by the centrifuge. The maintenance on the centrifuge was approximately the same as previously.

The next step in our experiments was to pump the underflow from the screen into a cyclone thickener which was mounted directly over the vibrating screen. This thickener increased the concentration of the solids to approximately 60 pct by weight and dropped the material back on the filter cake which had formed toward the discharge end of the screen. Unfortunately, the screen was not capable of handling this additional tonnage, and our experiments stopped at that point.

We have been considering installing a second screen to make possible the complete mechanical dewatering of this product by the use of the dewatering screen and the cyclone thickener.

Another possibility under study is to pump the underflow from this screen to a thickener which is available in the flotation plant, and to combine this thickened underflow with the original feed going to the screen. Again, however, a second dewatering screen will be required to handle the total tonnage.

* Lehigh Navigation Coal Co., Lansford, Pa.

O. R. LYONS*—I had an opportunity to read this paper ahead of the meeting, and I did a little pencil engineering on it. As Mr. Bertholf said, it is very difficult to make a comparison and to carry the results of work at one plant over to what might be expected at another. What I did was to find information on filtering operations more or less comparable to the type of operation that Mr. Bertholf has with his vibrating filter. The only information that I was able to find was for drum type filters, and I found the operating characteristics of the vibrating filter and the drum type filters were very similar. The moisture contents of the cakes were almost identical. The output per square foot was about the only way that I could compare their capacities—using square foot of screen area against square foot of filter area—and I found the capacity of the vibrating filter to be slightly greater per unit area than the capacity of the drum-type filters.

W. H. NEWTON†—Do I understand that the only escape for the solids is by overflowing the thickener? That is, does the filter have a chance to recover all the solids except that lost in the thickener overflow?

W. M. BERTHOLF (authors' reply)—Actually, the only escape from that part of the circuit is over the top of the thickener. There are other places the fines could be lost in the washery, but once they get into that part of the circuit, they must go over the top to escape.

* Battelle Memorial Institute, Columbus, Ohio.
† Jeffrey Manufacturing Co., Columbus, Ohio.

W. H. NEWTON—I would like to ask Mr. Lyons if, in the study of rotary filters, he has any basis for comparison of operating costs?

O. R. LYONS—No, I had no information on costs. The only information I was able to find was on screen size, moisture content, and tonnage output per unit area.

W. L. McMORRIS*—Are you wasting that overflow water or re-using it?

W. M. BERTHOLF—Right now, we are not re-using it.

D. R. MITCHELL†—What is the approximate per capita cost of one of these units?

W. M. BERTHOLF—It appears to be somewhere in the neighborhood of $200, for the screen.

W. H. NEWTON—The cost would be about $2500 for the complete unit including the vibrating power unit.

G. A. VISSAC‡—I do not like to come on the floor after I have been talking so long, but I thought you might be interested in our experience in dewatering, as well as drying our very fine coals. We have used both centrifuge and vibrating screens. The type of vibrating screens we have used in Canada are called the Zimmer. That is a screen of German construction, and I guess it is along the same lines as the dewatering screens you are using now. We use wedge wires, and the minimum size opening is a quarter of a millimeter. In our experience, the cheapest way is still a dewatering bin. A dewatering bin takes 48 hr to do work that takes 20 min in a dewatering screen. We use old wedge wire from our driers which we cover with brattice cloth, and

* H. C. Frick Coke Co., Pittsburgh, Pa.
† Pennsylvania State College, State College, Pa.
‡ Consulting Engineer, Vancouver, B. C., Canada.

in this way we have no losses.

The trouble with the centrifuge, as well as the dewatering screen, is that there is always an undersize, and the question is what to do with it? In some coals it amounts to a considerable tonnage. With the dewatering bin, we have no losses, and we arrive at about the same amount of remaining moisture, which varies according to the coal, from 10 to 16 pct.

So much for mechanical dewatering. Now for the drying. Some plants have to deal with that minus quarter millimeter. The practice in the old country is to treat those sizes by flotation and recover the coal by filter presses or disk filters like those used in mineral separation. Those machines are very expensive, and the remaining moisture is as high as 24 pct.

So, in most cases, you end up, very often, with heat drying. The flash type used for this drying is not new. This is called the Buttner-Rema-Rosin in Germany, and that type of drier has been in use there for 25 years. You can find it described in some reports made by some British Intelligence as well as, I think, American Intelligence. A very good description of the flash type of drier was given at the International Congress in 1935 in Paris. It is a far more complicated type of machine. It included a classifier, and the large sizes were ground again until they were able to go through. That is the reason why that type of drier was never very successful unless for a coke or briquette plant, because the amount of fines is increased, and coal below a hundred mesh is very difficult to work with. So there is still room for research work. At the present time, we think that we can do with the screen type drier just as good a job as is done with the flash type. The advantage of the screen type is that the fines go through the screen and there is an ascending current of hot air to dry them. The large sizes remain on the screen and can be carried out separately. In this way, crushing the coal is avoided, and that is important when handling bituminous coal.

V. F. PARRY*,†—The authors describe an economical method of recovering fine coal from washery sludges and slurries. I enjoyed reading the paper and found the presentation very interesting.

It is reported that total operating costs per ton of dry coal handled are 4.75 cents. This indicates a cost of 6.6 cents per ton of water removed from the slurry or sludge, assuming that the vibrating filters receive sludge containing 50 pct solids and reduce the moisture to 78 pct solids in the cake. It is estimated that the cost of removing water from lignite by evaporation is about 75 cents per ton of water removed under favorable conditions when lignite costs $1.00 per ton. If coal costs 15 cents per million Btu, it is estimated that the cost for evaporating a ton of water from coal will be about $1.00. Therefore, the advantage of mechanical filtering over evaporation in this instance is on the order of 1 to 15.

The vibrating filter appears to have a moisture-reduction limit of about 20 pct in the cake. This will vary with the size of coal and the distribution of sizes in the cake as pointed out by the authors. The average moisture in the filtered cake is about 22 pct and it may reach 28 pct when treating slurries. Coals of 22 pct moisture are penalized in utilization. It is somewhat difficult to handle the cake and to distribute it evenly in a coke-oven mix. Furthermore, the weight and latent heat of the moisture in the cake subtract considerably from the heating value. Reduction of the moisture in the cake from 22 down to 1 or 2 pct may have several advantages both from the standpoint of handling materials and from efficiency in utilization. I should like to present some discussion on the problem of reducing the moisture by evaporation, since this appears to be the only feasible way to remove the water remaining in the cake after mechanical filtering.

We have experimented for some time with high-speed drying of low-rank coals. These fuels contain bed moisture ranging from 15 to 50 pct, but this moisture can only be removed by evaporation. We have found that moisture can be removed at maximum speed by drying in the entrained and fluidized state, employing high-temperature products of combustion. Fine coal in the form of filter cake containing up to 90 pct moisture can be flash-dried with medium-temperature gases, such as is done in the Raymond flash-drying system illustrated in Fig 1 of the paper. In this system the cake is mixed with a portion of dried coal and then dispersed mechanically by a fan in the hot gas stream. Since the moisture is principally surface moisture, the evaporation is extremely rapid and the capacity depends upon the rate of supply of hot gas. In this system the temperature of the hot gas is limited to about 1300°F to avoid damage to the fan.

Hot gases at 2000 to 2200°F, and possibly higher, might be used for drying the filter cake, employing the technique we are using on low-rank coals. In this system the hot gases are generated under pressure and jetted into the fluid bed. Mechanical agitation in the bed could be employed to break up the filter cake without the addition of dried material. The thermal efficiency of such a system is 50 to 75 pct higher than that of the system previously discussed because of the higher-temperature hot gas and the possibility of excluding recycled dried coal. The capacity of a drying column employing the entrained and fluidized technique should be about 2000 lb of filter cake per hour per square foot. The net heat required to dry the coal is 316 Btu per pound, and the overall thermal efficiency of the drying system should be 85 to 90 pct. These estimates are derived from our experience in drying lignite of 37 pct moisture.

It is indicated, from studies we have made on costs of drying low-grade coals, that the moisture in filter cake can be removed at a cost of $0.75 to $1.00 per ton of moisture removed.

* Work on manuscript completed February 1949. Not subject to copyright.

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Cyclone Operating Factors and Capacities on Coal and Refuse Slurries

By D. A. DAHLSTROM, Junior Member, AIME

DISCUSSION

(*A. C. Richardson and Charles C. Boley, presiding*)

W. E. BROWN*—In the operation of the cyclone, what factors have you found that will affect its results as far as efficiency goes; for example, if you have coming through the cyclone a pulp of 20 gpm, that has, say, 25 or 30 pct solids, and then from operational characteristics this pulp changes to 35 pct solids but it is still maintaining the same ratio of flow. How does that affect the cyclone?

D. A. DAHLSTROM (author's reply)—It has a definite effect, as you know from Mr. Sutherland's paper,[13] in that as you increase the solid concentration of your inlet slurry, you will naturally have an according effect on the overflow concen-

* Humphreys Investment Co., Denver, Colo.

[13] References 13 to 23 are at the end of this discussion.

tration. This, of course, assumes that the amount of addition possesses the same size distribution—in other words, the size distribution of the heavier load is the same as the lighter load. The cyclone will tend to operate with about the same efficiency even though there is an increase in the loading. A very rapid complete overloading of the cyclone is not usually experienced. However, it will undoubtedly tend to go into a transition discharge for severe loadings unless there is some method of changing the underflow diameter. In this case the recovery undoubtedly will be injured to a certain extent.

There have been methods advanced for the use of a rubber joint at the bottom of the cyclone in order to be able to change the diameter of the underflow at any one time. I have not done any work at all on that, so I cannot answer that question, but we believe it is extremely important that the slope of the cone be maintained constant all the way down to the underflow discharge. As soon as the flow pattern of the cyclone is disturbed, the air current, which we think is important for the maximum elimination of solids, tends to shut off.

W. E. BROWN—Usually in an operation, a constant volume of pulp comes through any process, but the solids can vary considerably.

D. A. DAHLSTROM—Again I will have to go back to Mr. Sutherland's data, and you will observe that it would vary during a day's time, especially as you get a build-up of fine solids in your system. We found that the variation was not too great, that is, we did not have to go into any change in dimensions at all. We simply allowed them to stay put, and we did get a certain amount of damage to our percentage of extraction, because we progressed more and more toward the complete overloaded condition. However, it was not serious, I believe, and you could set your underflow dimensions to take care of, say, your average, or slightly larger than average, operating condition. It probably would take care of your peak periods and also would not give you too much dilution in periods when below average load.

J. D. GROTHE*—Your work was done essentially on homogeneous gravity solids?

D. A. DAHLSTROM—That is right.

J. D. GROTHE—The work at the collieries would be on heterogeneous gravity solids. Can you tell us the effect in this case?

D. A. DAHLSTROM—When you have a range of gravity material that is ranging from coal of, say, 1.3 up to pyrite of approximately 5.0, you will naturally have a range of 50 pct points. In other words, each particular particle of gravity has its own particular 50 pct point. That is why we had to standardize on standard specific gravity material for theoretical work, because we would have an introduction of too complex a variable by use of coal or refuse slurries.

In our analyses we use the hydrometer method proposed by the Casagrande, which assumes an average specific gravity of that fraction, and I might add that the method of hydrometer analysis has been used by soil mechanics and civil engineers, and is considered about as valid as any subsieve method. We found that this, tempered with a 1.25 safety factor, would give a suitable prediction of 50 pct points.

W. L. McMORRIS*—You made the statement that the cyclone could be used as a vessel for separation for particles of different gravity. You showed in your slides a differential between the 50 pct point of your standard material and that of a higher gravity material. Have you ever worked out, either experimentally or mathematically, the difference in the ratio of concentration shown in the cyclone as compared to the ratio of concentration in hindered settling or in free settling?

D. A. DAHLSTROM—May I change your question around to this: Have we shown any effect on the 50 pct point as we increase the solid concentration on the feed—does that sound logical?

W. L. McMORRIS—No. Let me give you an example. You made the statement that there would be a different 50 pct point on pyrite as compared to 1.3 gravity coal. If you apply the standard hindered settling formulas to 1.3 coal and, say, 4.9 pyrite, would the ratio of concentration that you obtain in the cyclone be as good as that which you obtain in hindered settling?

D. A. DAHLSTROM—We made no tests on that because we were entirely concerned with complete solid elimination with relatively dilute suspensions, and we found, as indicated, that when we used Stokes' law without any hindered settling considerations and obtained our correlations, we did as well as free settling. I do not know what would happen with hindered settling.

H. F. YANCEY† and M. R. GEER†—The cyclone paper by Fraser, Sutherland, and Giese,[13] and that by Dahlstrom are of great interest to us because the original work on the cyclone as a thickener of coal slurry was done by the Bureau of Mines at its Northwest Experiment Station in Seattle. The first paper is especially interesting because it represents plant operation as distinguished from laboratory-test information. Dahlstrom's paper is concerned principally with cyclone design and provides information hitherto unavailable on the energy requirements of cyclones.

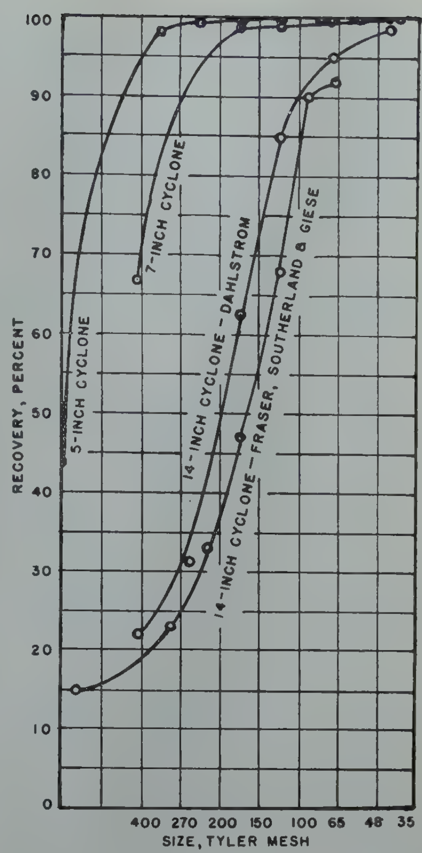

FIG 17—Comparison of solids recovery in 5, 7, and 14 in. cyclones.

One of the most important conclusions he has reached is that cyclone diameter has no influence on the efficiency of recovering solids; in other words, large cyclones are as efficient as small units. This conclusion, however, does not appear to be substantiated by published data on cyclone performance, even including that provided by his own work. To illustrate this point, Fig 17 shows the performance of 5, 7, and 14 in. cyclones, expressed in terms of the percentage recovery of the individual size fractions comprising the feed slurry. Data for the 5 in. cyclone represent test 82 from our own published report,[5] and that for the 7 in. cyclone are from test 2 of Dahlstrom's discussion[6] of this report. Two curves are shown for the 14 in. cyclone installed at Kayford, W. Va., one representing an average of 10 tests reported by Fraser, Sutherland, and Giese, and the other illustrating an average of 5 tests on the same unit reported by Dahlstrom. All of these tests were made on coal slurries, and all are comparable in that they represent vortex discharges containing less than 16 pct of the water present in the cyclone feed.

These curves illustrate clearly the superior recovery effected by the smaller

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cyclones throughout the entire size range present in coal slurries. Even in the coarser fractions, where the recovery of solids is relatively easy, the 14 in. cyclone makes an incomplete recovery. The importance of the superior performance of the smaller cyclones in terms of total solids recovery is illustrated in Table 5; here, the recoveries shown in Fig 15 for the 5 in. cyclone and for Fraser, Sutherland, and Giese's data on the 14 in. cyclone are applied to a slurry such as that treated at Kayford.

Table 5 . . . Comparison of Solids Recovery Effected by 5 and 14 In. Cyclones

Size, Mesh	Feed, Pct	Recovery, 5 In. Cyclone		Recovery, 14 In. Cyclone	
		Percentage of Size Fraction	Percentage of Feed	Percentage of Size Fraction	Percentage of Feed
Over 80	37.4	100.0	37.4	92	34.4
80 to 100	4.1	100.0	4.1	90	3.7
100 to 150	10.6	100.0	10.6	68	7.2
150 to 200	7.3	99.7	7.3	47	3.4
200 to 250	2.3	99.2	2.3	33	0.8
250 to 270	1.8	99.1	1.8	31	0.6
270 to 325	2.7	98.8	2.7	22	0.6
Under 325	33.8	55.0	18.6	15	5.1
Total recovery of solids			84.8		55.8

As shown, a 5 in. cyclone operating on this slurry would recover 84.8 pct of the solids in comparison with only 55.8 pct recovered by the 14 in. unit.

Thus, while from a theoretical standpoint it may be possible to operate large cyclones at velocities high enough to give the high recoveries characterizing small cyclones, in the pressure range commonly used the larger cyclones are distinctly less efficient. This argument should not be construed, however, to indicate that multiple small units are necessarily to be preferred to a single large cyclone. The cost of producing small cyclones equaling a larger unit in capacity may be greater, and in some instances the attempt to obtain complete solids recovery may not be justified economically. The advantage of the small cyclone is stressed here to correct any misconception about the relative efficiency of large and small units. Equipment manufacturers should consider the feasibility of producing small cyclones in multiple units for applications where maximum solids recovery is a justifiable objective.

Undoubtedly, Dahlstrom's conclusion about the relative efficiency of small and large cyclones was influenced largely by considering the 50 pct separation size, that is, the particular particle size divided equally between the two products, as a measure of the efficiency of solids recovery. Using this size as a criterion of efficiency, instead of using the information provided by the type of curves in Fig 15, is equivalent to characterizing the efficiency of a coal-washing process by the specific gravity at which the separation between coal and impurity is made, without any regard for the amount of heavy impurity entering the washed product or the amount of clean coal lost in the refuse.

Moreover, Dahlstrom's determination of the separation size is based entirely on a simple sedimentation method of estimating the size distribution of subsieve material. The dangers inherent in relying on such a method were pointed out in the reply to his discussion of our cyclone paper.[14] A concrete example of the errors involved in this procedure is provided by the data in Dahlstrom's test 6; complete data for this test were not included in his paper but were kindly furnished by him for the purpose of this discussion. In this test actual screen analyses showed the separation size to be about 100 microns, while the sedimentation-test data indicated a separation size of only 50 microns. This comparison is shown in Table 6, which shows the recoveries calculated from both screen analyses and sedimentation tests.

Table 6 . . . Recoveries Effected in Different Size Fractions, Dahlstrom's Test 6

Screen Analysis, Size in Mesh, U.S. Std.	Percentage Distribution		
	Feed	Overflow	Underflow
Over 4	100.0	0.0	100.0
4 to 10	100.0	0.0	100.0
10 to 20	100.0	0.0	100.0
20 to 40	100.0	0.5	99.5
40 to 60	100.0	1.4	98.6
60 to 100	100.0	6.5	93.5
100 to 140	100.0	29.7	70.3
140 to 200	100.0	53.3	46.7
Under 200	100.0	82.6	17.4
Sedimentation Analysis of Minus 200, Size in Microns			
Over 65	100.0	0.0	100.0
65 to 46	100.0	39.1	60.9
46 to 32	100.0	68.0	32.0
32 to 23	100.0	86.6	13.4
23 to 16	100.0	84.3	15.7
Under 16	100.0	92.0	8.0

Another anomaly is evident in the fact that sedimentation tests indicated that 100 pct recovery was effected in the sizes coarser than 65 microns, whereas the actual screen-analysis data indicated that complete recovery was effected only in the sizes coarser than 20-mesh, corresponding to 840 microns. Obviously there is no direct correlation between the equivalent diameters obtained by sedimentation tests and sizes determined by screening. With discrepancies of this magnitude inherent in the sedimentation method, it is clear that conclusions reached on the basis of its use must be regarded with great caution.

Another conclusion reached by Dahlstrom with which we are not in complete agreement is that the percentage of water entering the cyclone underflow has little bearing on the recovery of the finer solids. Actually, the data presented by Fraser, Sutherland, and Giese indicate clearly that, when the overflow orifice of the cyclone was restricted and hence a higher percentage of the water entered the underflow product, the recovery of solids was enhanced considerably. In fact, the common basis on which the performance of cyclones operating on different slurries can be compared readily is the distribution of the water to the overflow and underflow products; this distribution is controlled by the size of the cyclone openings. With a vortex-type underflow the actual percentage of solids in either the underflow or overflow products is dependent upon not only the percentage of solids in the feed, but also upon the size composition of these solids. Hence, comparisons based on thickening effect are not valid unless these factors are taken into consideration. The type of curves illustrated in Fig 15 represents the best means of comparing cyclone performance provided they represent similar percentages of the feed water entering the underflow product.

D. A. DAHLSTROM—Before commenting on Messrs. Yancey and Geer's discussion, it would be wise to first emphasize an important consideration of the original paper. The work was undertaken to obtain fundamental data and theory of the liquid cyclone which is of large importance in its fuller application and understanding. To achieve this it was felt that a special cyclone permitting individual observation of all operating and design factors would have to be constructed rather than use several different cyclones wherein all factors are simultaneously altered. Furthermore, as particle specific gravity is of major importance in such a separation process, industrial coal and refuse slurries which possess wide ranges in solid gravity could not be used. With this complex variable present, it would be very difficult to reliably determine individual factor effects. Therefore, a close gravity material was employed which would entirely eliminate this doubt. Accordingly, the experimental results and predictions can be expected to approach the ideal state and industrial performance may exhibit some deviation. However, fundamental studies are still of paramount importance in ferreting out the individual factors that effect any process operation which certainly cannot be reliably predicted from performance tests *alone* on industrial installations. (The emphasis is that the studies should complement each other.) In an effort to bridge the gap, the author has applied correlations of industrial data with experimental predictions so that the usual expediency of safety factors, where necessary, may be utilized.

For convenience of later discussion, reply will first be made to the objection of the hydrometer method for determin-

ing particle size. Any cyclone separation on minerals results in an appreciable concentration of the minus 200 mesh fraction. In fact, the 50 pct point (the validity of which will be discussed later) usually lies below the 75 micron size. Because of the proven unreliability of screen analyses below these dimensions, some subsieve method must be employed. The microscope cannot be used with sufficient accuracy for several reasons: (1) Only two dimensions are observed and a complex and exhaustive statistical analysis must be applied to determine a series of dimensions that portray size distribution. (2) If large ranges in solid specific gravity are present, it is almost impossible to correctly weight this factor. (3) The effect of particle shape cannot be reliably predicted. As particle shape is extremely important where surface area per unit mass is large (that is, fine sized particles), it would be misleading to determine elimination efficiencies based only on the linear dimensions of the particles and not consider particle shape or sphericity. (4) When materials are dried for microscope observations, it is difficult to prevent a partial, if not considerable, agglomeration of the individual particles, which introduces large errors.

From the above discussion, it is apparent that it would be desirable to utilize size analysis, employing liquid suspensions so that an equivalent particle dimension would be determined combining both particle size and shape. Settling velocity test procedures such as the hydrometer method have been found to be of considerable accuracy in this respect. The dependability of the test has been validated through exhaustive tests over the last 20 years and is now universally accepted by the soil mechanics and civil engineers. The proof and further development of Casagrande's original work[10] will be found in other works.[15–23] Today, these writers state the hydrometer method, when properly executed, is as accurate or better than any settling velocity method and generally will be appreciably simpler and time saving. Naturally, the accuracy will be decreased where large ranges in specific gravity are present. However, the tendency will be towards an averaging of specific gravity and all other methods will experience this same error.

Yancey and Geer, in Table 6 of their discussion, pointed out an apparent inconsistency in a hydrometer analysis; the screen analysis indicates a 50 pct point of about 100 microns while an hydrometer analysis on the minus 200 mesh fraction predicts 50 microns. This particular test run represents a severe deviation from normal operating conditions as a West Virginia cloudburst had completely muddied the natural streams used for make up water at this tipple. Solid distribution was altered and the ash concentration in the minus 200 mesh fraction increased from an average of 18 to 35 pct. Furthermore, as pointed out in the original paper, the underflow contained 58.3 pct solids, indicating a transition discharge. Thus, as the 50 pct point is a function of particle specific gravity, this point for the low ash coal was found at about 100 microns (exclusive of particle shape), while the high ash material point was located at 50 microns. This is further strengthened by the low ash concentration of 5.3 pct for the overflow 100 by 180 mesh fraction compared with 31.8 pct for the minus 180 mesh fraction. Finally, this was the only case of all the runs made, exclusive of complete overloading, that exhibited a 50 pct point for any of the material above the 200 mesh size.

Further objection was made (in their discussion) to the fact, "that sedimentation tests indicated that 100 pct recovery was effected in the sizes coarser than 65 microns, whereas the actual screen-analysis data indicated that complete recovery was effected only in the sizes coarser than 20 mesh." It must be remembered that hydrometer analyses were performed only on the minus 200 mesh fraction and extrapolation to the coarser sizes was not intended. Because there is a large range in specific gravity in this fraction with most of the heavy ash and pyrite material present in the underflow stream, hydrometer analysis on the underflow will tend to show a larger initial amount in the near 200 mesh size as one is forced to use an average specific gravity. By the same token, this near mesh material will appear to be absent in the overflow and thus will yield the anomaly. However, determinations at the 50 pct point for the *average specific gravity material involved* were proved to be dependable within 25 to 30 pct error, of theoretically predicted values. On the other hand, extrapolation of screen analyses in the minus 200 mesh region would be very dubious.

Considering the effect of cyclone diameter on degree of solid elimination, the author finds himself in disagreement with Fig 15 of Yancey and Geer's discussion. The portion of the curve below 270 by 400 mesh for the 5 in. cyclone and that below 140 by 200 mesh for the 7 in. cyclone was obtained entirely by extrapolation. This means extrapolating a curve established only from 100 down to 98.3 pct in the former case and to 98.9 pct in the latter, making any extension subject to considerable error. The curve representing the author's data on the 14 in. cyclone included 4 runs operating with a transition type underflow discharge in the 5 run average which were originally maintained to be inferior. The fifth run operating with a complete vortex discharge was definitely superior (run 3, Table 1 in the paper) with a 76 pct recovery of the total feed solids. This method of plotting is of course excellent as long as the same material is always processed. However, it does not show the effect of particle gravity on the resultant size distribution for prediction of results with other slurries. The author has always found that plus 200 mesh material in the overflow is very low in ash content (2 to 5 pct) when compared with similar sizes in the underflow. Finally, the character of the solids used with the 7 in. cyclone was considerably different than that of the 14 in. Ash content of the former was appreciably larger and the minus 200 mesh material was primarily silt and clay, the particle size of which is located near or well below the cyclone 50 pct point. By contrast, the latter material contained relatively small amounts of clay and silt. Undoubtedly the Washington coal slurries used with the 5 in. cyclone would exhibit singular characteristics and particle gravity differences. Accordingly, estimation of results from one cyclone to another without consideration of particle gravity or shape, especially in the minus 200 mesh region, will be affected by these factors.

Yancey and Geer show a predicted recovery for the 14 in. cyclone of 55.8 pct compared with their value of 84.8 pct on the 5 in. cyclone based strictly on size distribution without regard to particle shape or gravity. Thus their calculation ignores the difference in character between the two slurries on which the data are based, especially in the minus 200 mesh region. Furthermore, comparison upon their same basis using only run 3 (Table 1 of the paper) instead of the admittedly inferior average values which include 4 transition discharge runs would yield a predicted solid recovery of 72.1 pct. Run 4 of the same table which operated with almost the same moisture content in the underflow as the original 5 in. cyclone run quoted by Yancey and Geer would have a solid recovery by their method of prediction of 76.6 pct. This would appear by their own basis to greatly lessen the significance attached by Yancey and Geer to cyclone diameter.

The author must admit that he is largely influenced by the 50 pct point measurement in his statements regarding cyclone diameter. For theoretical work this method appeared favorable as it does portray the lower limit of size concentration efficiency. Regardless of what the size distribution of the particles above this point in industrial installations under normal operating conditions, the author has never encountered any significant concentration of material below this size. However, I am certainly willing to admit that cyclone diameter may have appreciable influences on the distribution of particles above the 50 pct point and I did not intend to give the opposite impression. Decrease in cyclone diameter may tend to decrease the percentage of particles above the 50 pct point found in

Table 7 . . . Actual and Theoretical Solid Recoveries for High-water Distributions to Cyclone Underflow

Runs 3, 4, and 5 of the paper

	Run 3	Run 4	Run 5
Total solids to underflow, pct	76.0	80.4	83.3
Total water to underflow, pct	11.8	23.9	30.1
Overflow water, run 3, now reporting to underflow, pct	$\frac{23.9 - 11.8}{100 - 11.8} \times 100 = 13.8$		$\frac{30.1 - 11.8}{100 - 11.8} \times 100 = 20.7$
Theoretical solid recovery increase over run 3, assuming no concentrating effect	$13.8(100 - 76.0) = 3.3$		$20.7(100 - 76.0) = 5.0$
Theoretical solid recovery, pct	$76.0 + 3.3 = 79.3$		$76.0 + 5.0 = 81.0$

the overflow. At the same time, the author believes the effect on the 50 pct point will be relatively insignificant. It should also be cautioned that recent work at Northwestern has indicated that cyclone retention time, percentage of cyclone radius covered by overflow and inlet nozzles, length of cylindrical section, shape of cyclone top, and position of the overflow point with respect to the inlet and top of the cylindrical section may be of appreciable importance which may further minimize the significance of cyclone diameter.

In regard to percentage of water entering the cyclone underflow, the author also believes that it does have a bearing upon the final recovery of solids. However, it is maintained that this effect is insignificant after a complete vortex discharge has been attained. Proof of this will be found in the results of runs 3, 4 and 5 of Table 1 in the paper. Run 3 possessed a normal vortex discharge recovering 76 pct of the solids with a water split to the underflow of 11.8 pct. Runs 4 and 5 were made with severe increases in water distribution to the underflow caused by the insertion of smaller overflow nozzles. Assuming that increasing volume split to the underflow has no concentrating effect on the solids and this excess water carries with it only those solids in the same dilution found in the overflow, a theoretical calculation for solids recovery in runs 4 and 5 can be made as shown in Table 7.

The slight deviation of theoretical values (assuming no concentrating effect) from actual results may be due to the test error, the smaller overflow nozzles and their advantageous effect on the correlating factor $\frac{(gpm)^{0.53}}{(be)^{0.68}}$ of runs 4 and 5, or a beneficial influence on the near 50 pct material. In the latter case, solid material which is very close to the 50 pct point will tend to concentrate largely in the inner cyclone spiral. Thus by directing more of this flow pattern to the underflow, some concentrating action will result. However, it will usually be small and at the expense of a higher moisture content of the underflow product.

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Application of Screening and classification for Improved Fine Anthracite Recovery

By W. J. PARTON, Member AIME

DISCUSSION

(*A. C. Richardson and Charles C. Boley, presiding*)

D. R. MITCHELL*—The Chairman mentioned that we have had many papers on cleaning of fine coal and treatment of wash water solids. There are, of course, two reasons for that. One is that we have legislation, particularly in the east, that is making us go into these things whether we want to or not.

It just so happens at the present time that I have four plants that are going up under my general direction, three new and one rehabilitation job, and in all of them, we have these problems. Therefore, I was very happy to be able to listen to these papers, because some of the things that are in them are going to help me solve some of the problems that I am facing.

Of course, the second reason why we are concerned about fine coal recovery is the fact that coal has become a luxury item in a good many places, particularly in the anthracite region, so that we cannot afford to have it go to waste like we did in former years.

There is a third minor reason, and that is that we are finding how to handle and use the extremely fine sizes that we did not know how to use in former years.

Now, I have a couple of questions that I would like to ask. First, what are the maximum solids that you could have in a pulp going to one of these launder screens? I realize that it would be for anthracite. Second, of course the life of these screen surfaces is not very long—with brass or bronze, I believe it is a week, and with stainless steel, a month. Would it be possible to use screen surfaces like the Bixby-Zimmer round rod for the small sizes and possibly increase the life? Have you experimented with that?

Another point that I would like to bring out, both in Mr. Parton's paper and in some of the others, in studying cyclones and any settling apparatus where we use orifices, the life of the orifices is very short, and that is a problem that still has to be solved. The capacity of the cyclone seems to be small, and if you have a bat-

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tery of them, there is one plant going up in central Pennsylvania that has a battery of cyclones, and if you have to have an attendant around those cyclones most of the time watching orifices, it is not so good.

At this meeting we have talked quite a lot about cyclones. There is another piece of apparatus that has been in commercial use on coal for several years, and it was mentioned, I believe, at the New York meeting, but we have had very little mention of it. That is a Vortrap. It was developed in the paper industry, and the recovery of solids is also by centrifugal force. It does seem to have a characteristic of not having quite as high abrasion, and the two that I know of that are in use also have much greater capacity. They are recovering solids from typical sludge tank overflows to a percentage as low as or lower than cyclones would in the same circuit. It is an apparatus that I think should be looked into by people having this problem. Of course, we do not know all about it. It needs a great deal of work on it, but it does not seem to have nearly the abrasion at the orifices that there is with cyclones.

W. L. McMORRIS*—As Mr. Parton said, there has been a lot of talk about classification of table feeds, and very little done about it. I think Mr. Yancey had something to do with it about twenty years ago along with Byron Bird, and the way was not entirely uncharted, but was full of reefs.

We did have a very tough coal problem. We had a high sulphur coal. We wanted to get an elimination of pyrite. We had to consider it as an ore dressing problem to recover the pyrite to make a metallurgical coal out of a high sulphur field. As far as cleaning the fines, it was further complicated by the fact that the coarse middling contained an abnormal amount of pyrite. The sulphur recovery curve plotted similarly to an ash recovery curve on a sink float basis showed a terrific hump in it between about 1.35 and 1.55 sp gr.

Experimental crushing indicated that that curve could be flattened and brought down to a normal curve if we were to crush that coal to pass ¼ in. and treat it by some other efficient method. We were afraid that ordinary tabling of fine mesh coal to which had been added a very substantial percentage of crushed middling would be impossible unless we took a leaf out of the ore dresser's book and used classification.

With some little help from us, a classifier was devised. It was an open bottom classifier, using a gentle vortex action in the rising column and a pinch valve for withdrawing the solids from the bottom of each cell. The pinch valve was designed by the Bureau of Mines for a single cell classifier and elaborated on for an eight cell unit which we put in. We have found that the classifier has done all and more than we expected of it. We are able to take a very respectable refuse from the first two cells without the tabling of those two at all. The products of the other cells go to tables and are discharged at a sufficient density that we have to add push water in order to get proper table feed density, which is a big advantage, and I think its lack was one of the things that was the cause of the failure of your work twenty years ago.

The overflow of the classifier is another problem. We still have to recover the minus 28 mesh overflow which we are doing through a large thickener and bringing that back to an additional battery of tables.

Our plant is brand new. We do not have very much in the way of metallurgical results. The size spread in the classification from the low gravity coal to high gravity pyrite is pleasingly broad, which makes the table separation, especially of the natural ¼ by 0, very good. We do have a lot of middlings on all of our tables, middlings which would still be middlings if it were crushed to pass 325 mesh, so we are having to make our cut in the middlings.

We have sink in our float coal and float in our sink. We have not yet determined where the economical point of that is going to be. We have to get together with our principal customer, Carnegie-Illinois Steel Corp., to find where we are going to make the cut. We do not know yet how much tonnage our tables will handle with this classified feed. We have tried to bog them down and have not been able to do it yet. The coarse coal end of the plant does not screen out the coal fast enough for us to over-load the tables.

We anticipated when we built this plant that our fine coal plant was going to be the bottleneck—the heavy media plant and the fine coal section are two separate and distinct plants as they now stand—but, much to our surprise, 75 pct of our table plant walks along with full rated capacity on the heavy media plant. Now we have to do something with the heavy media plant to make it catch up to the table plant.

But the classification of a table feed for a tough coal certainly has a lot of things to recommend it. Whether or not that could be used economically on a more simple washing problem is something else again, but for a tough one, we feel that the classification is going to pay off.

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H. F. YANCEY*—This whole session has been extremely interesting to me, and I hope it has been to all of us. The papers have been of a high calibre, and so has the discussion.

Mr. McMorris's statement about the classification of the feed to coal washing tables is certainly interesting. We had the trouble that he spoke about in the work that I did with Byron Bird some years ago, that of having too much water in our classified products. They had to be dewatered ahead of the tables. But the thing of interest to me in his remarks is that he found it hard to overload the tables when using a classified feed, which is the very thing that we found. We had to dewater ahead of the table. Mr. A. C. Richardson, too, worked on this study at Seattle.

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The further remark I would like to make concerns Mr. Parton's paper: I should like to compliment the Lehigh Navigation Coal Co. and Mr. Parton for releasing the information about this launder type screen. I feel that it is going to have great usefulness in the coal preparation industry.

W. J. PARTON (author's reply)—In the first place, answering Dr. Mitchell's questions, I do not know what the maximum percentage of solids in the feed could be to have the launder screen operate efficiently. I feel, however, that the maximum percentage of solids will be determined, to a certain extent, by the amount of oversize material in the feed. Since most of the water is lost through the deck, the percentage of solids in the oversize product will increase. Because the launder screen operation is based on having sufficient water to convey the oversize solids over the screen deck, sufficient water must be supplied either in the feed or at some point along the screen deck to do this.

An interesting application we are now planning to make with this screen is to make a 28 mesh cut in the feed discharge from the diaphragm pumps at the Tamaqua Colliery flotation plant. This feed product is being discharged into the plant at approximately 35 pct solids by weight. Just how this is going to work out, we do not know, but the only thing involved in the experiment is the time involved in having some carpenters make the launder screen. In this particular case, I believe we will find the use of some dilution water at the discharge end of the screen necessary to push the oversized particles from the screen.

Dr. Mitchell also asked about the possible use of Bixby-Zimmer cloth or wedge bar screen on the launder screen. Unfortunately, these types of screen cloth are not satisfactory. A square mesh cloth must be employed having approximately 50 pct open area, to prevent blinding. We have obtained what we feel is very good life with stainless steel cloth. I do not believe I mentioned it when I presented the paper, but the life of the stainless steel cloth is 450 hr as compared with 80 hr for bronze cloth.

Launder Screens

J. S. JOHNSON*—The cleaning and

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screening of fine anthracite, as Mr. Parton admits, is a difficult problem. It must be attacked in the field as circumstances present themselves. There is no iron clad rule that can be adopted.

I want to refer mainly to the launder screens which were first tried out at Nesquehoning Colliery. There is a very old saying, "Necessity is the mother of invention," and that was the case at Nesquehoning Colliery. The amount of water and fines to be handled and limited space between cleaning machines presented a problem of either installing vibrating screens and pumps or some other method of handling the large amount of water plus the fines flowing with the water.

It was here that Mr. Parton conceived the idea of the launder screens and the results, as shown in Table 2, are very good. Comparing Table 1 with Table 2, you will notice the feed tonnage for both tables is 60 tons per hour. The coal production is 29.8 tph for Table 2 and only 5.5 tph on Table 1.

This is a step in the right direction, to focus our minds and thoughts to stationary implements wherever possible. It is to be admitted that we are living in a machine age, but machines require rigid foundations and power to operate, along with maintenance of moving parts. The only maintenance on the launder is the stationary screen.

Fig 1 gives three views of this launder screen. The discharge orifices are really novel and interesting and actually act as a feeder when used between the different size cleaners, No. 4 and No. 5 buckwheat. In other words, these orifices help to take care of the surges which come intermittently in our preparation plants. As Mr. Parton has said, this type of screen can be used to remove excess water from a feed product ahead of cleaning equipment as well as removing the high ash fines from the feed material.

Fig 3 shows the set up of two laundry screens, one 3 by 18 ft before the cleaner and one 6 by 6 ft launder after the cleaner.

Fig 5 shows two launders in parallel with each launder having different mesh screens, the first one 16-mesh delivering No. 4 buckwheat over screen, and the second launder with 24-mesh screen delivering No. 5 buckwheat, each one to their respective cleaning tables.

It is to be understood that a complete study should be made regarding installation of stationary screens, tests procured giving amount of water and solids to be handled; also, very important is the analysis of the solids and of the ash by sizes. To get the best results the pitch on which the launder is installed will vary according to the amount and character of material to be handled.

Table 3 and 4 give results of tests made on parallel launders. The undersize to waste of the solids shows 8.84 pct with composite ash analysis 38.91 which shows there is some good material going to waste.

Fig 7 shows orifice design. Noting what Mr. Parton has to say about the length of service this is giving, I will suggest that rubber be tried out in the orifice as it has been my experience that rubber outlasts all metals and even glass, when properly installed.

Fig 8 shows oversize classifier. This is another interesting feature which Mr. Parton explains very thoroughly and will, no doubt, be improved upon in the near future.

All in all, I consider this paper a very excellent one and wish to compliment Mr. Parton on the way he has outlined the tests, figures and charts along with graphs and photographs.

Aerial Photographic Contour Maps for Strip Mines

By GEORGE HESS and R. H. SWALLOW, Member AIME

DISCUSSION

(*L. C. McCabe and Robert P. Koenig, presiding*)

C. G. BALL*—These maps are obviously quite helpful in many types of mining engineering, but I want to find out if the prints which you obtain in the first step toward making any aerial contour map have already been corrected for tilt. Is that done afterwards only if you are going into the process of making a contour map?

GEORGE HESS (authors' reply)—The tilt correction is the first step in either compiling the mosaic or compiling the total map. They must be corrected.

C. G. BALL—But are the contact prints themselves corrected for tilt?

GEORGE HESS—They are not corrected.

E. R. KAISER†—I enjoyed this paper very much, and am glad to learn that the methods have a high degree of accuracy and are in use in the coal industry. Accuracy in dimensions is understandable from the paper. Would you tell us how accuracy can be obtained in the vertical dimensions by means of the photographs so as to permit drawing contour lines?

GEORGE HESS—To explain our process, even in a limited way, is quite a large order. However, the basic principle involved is the measurement of parallax by viewing two overlapping photographs stereoscopically. It would almost be necessary to have the various instruments here on view to adequately explain their operation.

E. R. KAISER—Are the elevations for reference obtained at more than four points?

GEORGE HESS—The pattern of control is four points obtained in the corners. Primarily, that is to restitute the picture and horizontalize it. The vertical control points are read in on the stereometer and a direct comparison is made while drawing the contour lines. You do have to have those four points. But one stereoscopic pair of 5 ft contours has some 340 acres on it, consequently for every four points picked up 340 acres are contoured.

E. R. KAISER—How does your method compare with that used by the military reconnaissance during World War II?

GEORGE HESS—The Army did not get into this refined mapping. Most of their work has been reconnaissance mapping by taking one vertical and two oblique views. Basically, the same principle is used. Our maps are for engineering purposes. The Army was primarily interested in reconnaissance.

E. R. KAISER—You could take more than 48 pictures in one flight?

GEORGE HESS—Yes, the Brock camera accomodates 48 pictures per magazine. We are limited by the hours of greatest light, though, 2 hr before high noon, and 2 hr after high noon. Shadows do affect the picture. The shadows can-

* Paul Weir Co., Chicago, Ill.
† Bituminous Coal Research, Columbus, Ohio.

not be too long when contouring.

GEORGE ASHLEY*—One thing you did not mention is that if you do not quite get things on the map, you can turn back to the photographs instead of having to go back to the field.

GEORGE HESS—That is a very good point.

GEORGE ASHLEY—As a matter of fact, having used both photographs and maps made from them for several years now, I have, in recent years, used the photographs for field maps, and have made my station points on the photographs and then transferred them to the map. You can read the details much better, particularly if you use overlapping photographs, and the stereoscopic pocket lenses with them.

R. P. KOENIG†—I would like to emphasize the point Mr. Hess touched on. From the point of view of top management of a coal company or any other company, aerial mapping has great advantages. In the wintertime, one is sometimes hesitant in sending the engineers out to do topographic mapping, while one does not hesitate to send an airplane out. This gives the advantage of doing a job quickly without having to consider where you will get the ground engineers. Particularly if they are tied up on some other work, and it affords flexibility from the management point of view. This we found to be of great advantage. I think those of you who try this method further will come up with the same results.

* Consulting Engineer, Harrisburg, Pa.
† Ayrshire Collieries Corp., Indianapolis, Ind.

W. B. ROE*—These remarks are more a supplement to Mr. Hess' paper than to a discussion in that our use of aerial photographic maps has been entirely limited to use of existing photographs flown by the several Federal Government agencies such as the Soil Conservation Service, U. S. Geological Survey, Corps of Engineers, T.V.A., and others. Also we have had no outside work done on interpretation of the photographs, but have done our own plotting, and so on, with some very little outside consultation with people familiar with this kind of work. Consequently we have not produced or used any maps of the quality or accuracy described by Mr. Hess.

We first became interested in using aerial photographs in connection with our prospecting for lignite in North Dakota, principally because we needed reliable base maps of areas where no maps, other than small scale Land Office maps or state road maps were available.

Our first use of these photographs was to construct reasonably accurate maps on which to plot our prospect holes, land holdings, drainage, and so on, in these North Dakota areas. Little use could be made of the photographs in this area in mapping geology as the relief is quite low and the whole area is covered by surficial deposits which mask the actual bed rock except in very rare cases. They have been very helpful, however, in correcting property lines, locating drainage lines, divides, and other surface features. Use of stereoscopic pairs has also been quite helpful in tracing drainage and divide lines even though no actual elevations or topography have been taken from them.

* Truax-Traer Coal Co., Chicago, Ill.

Our use of aerial photographs in Illinois has been about the same as in North Dakota with about the same limitations due to similarity of terrain. However, they have been used to a lesser degree here because of the greater availability of recent fairly accurate topographic quadrangle maps and because our own property already had been pretty well covered by our own ground survey maps.

In West Virginia and other parts of the more rugged Appalachian coal fields we have used these photographs to quite some extent as in North Dakota and Illinois and also to a much greater degree as stereoscopic pairs in examining the geology of the region, as in these unglaciated areas the bed rock sequence and structure show up much more plainly in the photographs.

The stereoscopic pairs have been used for laying out tramroads, crop-line stripping, incline locations, and other uses, as looking at the pairs is second best only to a field examination of the area, and often discloses facts not discovered in the field, because of the impossibility of seeing the area as a whole when on the ground.

As stated in opening, this is hardly a discussion of Mr. Hess' paper, but does, I believe, show the possibilities of using the aerial photographs already in existence to facilitate prospecting and development of strip coal properties.

Coal Mine Development in Alaska

By ALBERT L. TOENGES, Member AIME

DISCUSSION

(*Richard H. Swallow and B. W. Dyer, presiding*)

C. P. HEINER*—I would like to ask Mr. Toenges about the highest rank coal. I did not get that clearly. What kind of coal is that?

A. L. TOENGES (author's reply)—The coal in the Matanuska field ranges from high volatile "B" bituminus to anthracite. It all depends on which direction you go in the Matanuska field.

C. P. HEINER—Is it a large reserve?

* Utah Fuel Co., Salt Lake City, Utah.

A. L. TOENGES—That is what we hope. That is what we are trying to find out.

B. W. DYER*—In those strip beds, I wonder how far back the coal would weather?

A. L. TOENGES—I never noticed particularly.

B. W. DYER—The weathering does not extend back like it does in this country?

A. L. TOENGES—No, it does not.

* U. S. Geological Survey, Salt Lake City, Utah.

B. W. DYER—Is that due to the freezing?

A. L. TOENGES—Probably so.

B. W. DYER—In this country, you would expect nothing but weathered coal.

R. H. SWALLOW*—Along that line, you mentioned some of the seams were on fire. Did you ever know how far back they burned, or was it hard to tell?

A. L. TOENGES—The fires occurred in the mine. That one you refer to was from an inside fire.

* Ayrshire Collieries Corp., Indianapolis, Ind.

H—Industrial Minerals

Beneficiation of Industrial Minerals by Heavy Media Separation

By G. B. WALKER and C. F. ALLEN, Members AIME

DISCUSSION

Why Differential Density Separation

K. F. TROMP*—In dealing with the question of the most suitable kind of solid media for heavy density suspension processes Walker and Allen point out that the particle size of the solid media should not be taken too fine, as the viscosity increases with the area of the solid media and a low viscosity is essential for high tonnage and accurate separation. A coarser particle size of the solid media will, in their opinion, of necessity give rise to a differential density in the bath (higher gravity at the bottom of the bath than at the top) but they advocate acceptance of the differential density rather than a higher viscosity.

Though I fully agree with the choice the authors have made, I cannot subscribe to their view that only by accepting a differential density in the bath a coarse particle size of the solid media can be used. There certainly is another alternative: stronger agitation. Applying sufficiently strong vertical currents, a uniform gravity can be obtained quite well in a suspension of a coarse solid media. Of course, this is not a very attractive solution, for it means a degradation of the true gravity separation and a step backwards to hydraulic classification, which makes the washing dependent on size and shape of the particles.

However, to a greater or lesser extent, this is what actually takes place in all the heavy density suspension processes relying on a uniform gravity in the bath. The so-called "stable" suspension processes make no exception. They all "stabilize" their suspensions by introducing or creating vertical currents, be it upwards or downwards or both, be it by hydraulic or by mechanical means. In fact, there is no such thing as a "stable" suspension in gravity separation, as the very reason for the use of suspensions in this field is the property that the solid media is able to settle and so facilitate the recovery.

I have been enlarging on this point because the characteristics of the various processes can only be well understood and viewed from the same angle (from Barvoys up to Chance) when the fact is recognized that mechanical or hydraulic agitation is a condition *sine qua non* for obtaining a uniform density from top to bottom in a suspension.

Is a Cone-shaped Vessel Essential?

Of the two alternatives for getting a low viscosity Walker and Allen have preferred correctly the sacrifice of uniform gravity in the bath instead of increasing further their vertical current and agitation. The resulting differential density of the bath brings the problem of how to prevent accumulation of intermediate gravity products in the bath, an accumulation which, if not prevented, would ultimately plug their cone. According to the authors an open-top cone combined with a downdraft current of the bath liquid would be the only suitable way to cope with such suspensions and they assume as a fact that "in any vessel other than a cone, such a differential density could not be tolerated."

My experience is quite different. In my process, which has been in successful operation for more than a decade, differential density of the suspension is applied ranging from values below 0.1 up to differentials above 0.5, according to the prevailing requirements of the individual plant. In this process, which is characterized by the use of horizontal currents in a suspension of differential density, the form of the vessel is of secondary importance and different types are in operation. It so happens that *none of these are in the form of a cone.* The fact that 24 washboxes on my process have been installed and 12 others are under construction may constitute sufficient proof against the opinion that only a cone-shaped separator would be suited for differential density separation.

Horizontal Currents in Differential Density Separation

I myself have some doubts as to the suitability of a cone with downdraft for dealing with differential density (or, for that matter, any other washbox relying on vertical currents for removing the intermediate gravity products). It appears to me that it is restricted to feed of small size only and even then with watchfulness. If we take, for example, a piece of 2 in., the draft necessary to pull such a piece down to a zone wherein the density of the suspension is, say, 0.03 higher, is quite considerable. For a suspension of, say, 1.6 sp gr the downdraft will have to be in the region of 3 in. per second.

Unfortunately, most of the differential in density is in the part immediately below the reach of the top current which transports the floats. Consequently, we need the downdraft where we like it least: in the upper part of the cone. This entails the risk that light float particles are carried away with the downward current. This current of, say again, 3 in. per second would carry particles up to 1.3 sp gr and ⅜ in. size into the 1.6 gravity zone. This is prohibitive. It is also prohibitive because a downdraft of 3 in. per second in the upper part of the cone would require a tremendous circulation of medium. Half way up a 20 ft diam cone, a downdraft of 3 in. per second would correspond with 8500 gpm.

With the downward current following the way of least resistance, the strength of the downdraft will not be exactly the same at different places of a cross area. If, as I anticipate, the center of the cone is favored, the strength of the downdraft will fall below the critical value near the

* Consulting Engineer, Kerkrade, Holland.

periphery and give rise to a ring of stagnant intermediate gravity products, unless still larger amounts of suspension are circulated.

This example may give an idea of the restrictions which have to be observed when trying to remove the intermediate gravity products from a suspension by means of vertical currents. It explains why only a small differential can be allowed when material other than small sizes are washed. A small differential in density means of necessity that either high viscosities or strong agitation have to be accepted. There is no way to get around the law of Newton.

In my process, conditions are quite different. The intermediate gravity products are removed horizontally instead of vertically. In other processes, the strength of the currents is all-important, as they have to oppose (upward currents) or to neutralize (downward currents) the effect of the gravitational forces on the suspended products. In my process, the suspended products are shoved, not lifted. They remain in a zone having a specific gravity equal to their own. This explains why a small velocity of the horizontal currents is sufficient. Whether the differential in density between the top and the bottom of the bath is small or great makes therefore no difference in the efficiency and reliability of the removal of the intermediate gravity products. Nor does the size of the products do so.

For further particulars on my process I would refer to the literature[1,2,3] and especially to the latest paper, written by Holmes,[3] who gives a clear analysis of the problems connected with heavy-media separation.

References

1. O. Schaefer: The Tromp Dense Medium Process for Coal Washing. *Colliery Guardian* (1938) No. 4068, 1073–1077.
2. C. W. H. Holmes: Notes on Specific Gravity Washing with Special Reference to the Tromp Process. *Colliery Guardian* (1939) No. 4120–4121. *Trans.* Inst. Min. Engr. (1939–40) 98 (5) 175–207.
3. C. W. H. Holmes: Notes on Coal Washing with Dense Media. *Colliery Guardian* (1948) No. 4563, 856–859.

G. B. WALKER and C. F. ALLEN (authors' reply)—The discussion by K. F. Tromp makes some assumptions with respect to the currents in heavy-media separation cones which are not borne out by actual operation on a commercial scale. The assumption is made that a very large upward rising current of the order of 3 in. per second would be necessary. Upward rising currents of very large magnitude are indeed encountered in cones using the Chance process, where the medium solids are relatively coarse and a very strong upward current of water is necessary, resulting in a separation which is correctly described as resembling, in some respects, a separation effected by hindered settling. In practical operation, in cones using heavy media, there are no vertical currents of comparable magnitude. The medium has so low a settling rate that the gentle stirring effected by medium return and removal of sink and float products maintains the suspension of medium solids. From a purely theoretical point of view, no suspension is absolutely stable unless the particles are all of colloidal size. However, the stability of the normal medium used in heavy-media separation processes requires such negligible agitation as compared to processes such as the Chance process that, for all practical purposes, the medium may be considered as stable, even though it is true that if the medium stands for enough hours, or days, it will slowly settle. The practical meaning of the expression "stable" was the one used in the paper.

Mr. Tromp assumes that unless a very strong downward current of the order of 3 in. per second exists, heavy-media separation cones would plug up with products of intermediate density. In practical experience in 43 commercial heavy-media separation plants treating a large number of different ores, and with a total hourly capacity measured in the thousands of tons, this problem is normally not encountered, even when treating material as coarse as 9 in. and as fine as 10 mesh, and the heavy-media separation cones are normally operated with an agitation which does not produce upward currents of any significant magnitude. Neither excessive viscosity, nor violent agitation have proved necessary in commercial practice to prevent plugging of heavy-media separation cones with particles of intermediate gravity.

To illustrate the ability of a cone type separator to handle near gravity material without difficulty we show in Table 1 the heavy liquid analysis of a coal recently cleaned on a continuous basis.

Table 1 . . . Heavy-media Feed

Product	Weight, Pct			±0.015 Sp Gr	±0.03 Sp Gr	Ash Assay, Pct		
	Indiv.	Cum. Float	Cum. Sink			Indiv.	Cum. Float	Cum. Sink
Calc. raw feed	100.00					14.00		
Float 1.32	32.24	32.24				7.43	7.43	
1.32 × 1.33	10.98	43.22	67.76			9.82	8.04	17.12
1.33 × 1.34	10.11	53.33	56.78	29.84		11.08	8.61	18.53
1.34 × 1.35	8.75	62.08	46.67	25.23	41.30	12.33	9.14	20.21
1.35 × 1.36	6.37	68.45	37.92	20.21	33.33	13.55	9.55	21.96
1.36 × 1.37	5.09	73.54	31.55	14.47	25.22	14.51	9.89	23.65
1.37 × 1.38	3.01	76.55	26.46	10.10	20.04	15.24	10.10	25.41
1.38 × 1.39	2.00	78.55	23.45	8.58	15.73	16.14	10.26	26.72
1.39 × 1.40	3.57	82.12	21.45	7.63	12.30	17.38	10.57	27.70
1.40 × 1.41	2.06	84.18	17.88	7.29	9.87	18.53	10.76	29.98
1.41 × 1.42	1.66	85.84	15.82	4.30	8.41	19.64	10.94	31.16
1.42 × 1.43	0.58	86.42	14.16	2.78	5.70	21.27	11.01	32.54
1.43 × 1.44	0.54	86.96	13.58	1.98	4.48	19.84	11.06	33.03
1.44 × 1.45	0.86	87.82	13.04	2.24	4.19	21.86	11.17	33.57
1.45 × 1.46	0.84	88.66	12.18	3.07	4.97	22.55	11.25	34.40
1.46 × 1.47	1.37	90.03	11.34	3.57		24.57	11.48	35.30
1.47 × 1.48	1.36	91.39	9.97			31.85	11.78	36.78
Sink 1.48	8.61		8.61			37.52		37.52

More recently an anthracite coal was treated in a continuous operation. The medium was adjusted so as to deliberately set up a high differential density. The specific gravity of the float medium was 1.65 while that of the sink medium was 1.95, a differential of 0.30. Of the total quantity of material between 1.65 and 1.70 sp gr in the feed less than 5 pct remained in the cone at the end of the test. This illustrates the ability of a constantly increasing downdraft in the cone to offset the teetering effect of even an extremely high differential density.

Mr. Tromp incorrectly assumes that the downdraft created by the removal of medium through the air lift will be reflected throughout the entire body of medium in the cone which is not the case. If a cone is discharging float over a weir carrying a 3 in. crest of medium we can safely assume that the actual zone of separation is not deeper than 5 in. If the medium discharged by the air lift is returned to the cone at a level more than 5 in. below the surface, there will be no downdraft in the separating zone. In fact if all or part of the float drainage medium is returned to an intermediate zone in the cone an updraft will be created toward the surface of the cone while a downdraft will exist from the point of medium return to the air-lift opening.

The Link-Belt float-sink concentrator referred to in our article and the Nelson L. Davis Co. heavy-media precision processor both employ a very mild rising current through the bath working in conjunction with a horizontal conveying current on the surface. The Akins separatory vessel operates with primarily a horizontal current having a slight rising component and employs a very high differential density. The highly successful operation of all four types of separatory vessels together with the fact that they differ so widely in their method of opera-

tion leads one to conclude that the success of heavy-media is due primarily to the development of a method for recovering and cleaning finely ground magnetic medium rather than to the manipulation of currents required by separating methods which attempt to make a sink-float separation in a hindered settling classification system.

Recent Trends in Asbesta Mining and Milling Practice

By MICHAEL J. MESSEL, Member AIME

DISCUSSION

(*J. L. Gillson and A. B. Cummins, presiding*)

W. P. MOULD*—Has consideration been given to the problem of retreating the 4,000,000 ton tailings pile to recover the very considerable amount of asbestos fiber that was lost over the years due to insufficient operation and poor recoveries? Also, what was known about the possible content of chromite and the sulphides of nickel and cobalt?

M. J. MESSEL (author's reply)—Consideration was given to the problem but nothing concrete has been done up to this time.

W. P. MOULD—I am certain the tailings contain a very considerable quantity of fiber, not all of which is in the shortest lengths, because in the milling of talc rock from a neighboring location having a common origin with the asbestos bearing serpentine source of the tailings pile, a concentrate carrying 15 pct Ni and Co in complex sulphides and constituting about 4/10 of 1 pct of the talc rock was recovered.

In the belief that the tailings also contained an appreciable content of chromite (then in demand, 1937) it was suggested to the management of the asbestos company that a truck load of the current tailings be sent to the flotation plant of the talc company where it would be put over the concentrating tables in an effort to discover the presence and roughly the percentages of chromite, nickel, and cobalt.

Preparations for the test were woefully inadequate and the test was completely snarled up by the surprisingly large quantity of asbestos fiber from 1 in. long down to very short fiber, all of which matted up and plugged launders and pumps with disastrous effects and practically the total loss of the sample.

If anything was indicated by the test, it was that a wet process for milling asbestos was worthy of very serious consideration.

M. J. MESSEL—A wet milling process for tailings retreatment only, is at present under study at our plant.

One asbestos mining company in Canada has constructed a fairly large test plant for this purpose, but as yet no definite results have been obtained other than in a small test unit plant which were in some respects quite encouraging.

* Rock of Ages Corp., Barre, Vt.

A New Incline in the Metaline District

By CHAS. A. R. LAMBLY,* Member AIME

Location

In the extreme northeast corner of the State of Washington, on the Canadian border, lies the Metaline mining district. This district is old in history, but young in production.

Geology

The Metaline district is a zinc-lead area of the replacement type in dolomite and limestone. The ore bodies of the Josephine horizon are in many ways similar to the ore bodies of the famous Tri-State zinc fields. The beds are faulted and folded and have varying low dips in varying directions, and underlie large areas of the district.

History

Production started in 1927 on a very limited basis. The property is now mining and milling 700 tons per day. The mine is opened by adit tunnels and a vertical shaft. As the ore horizons gained depth, it was necessary to sink inclines to follow the ore horizon (see Fig 1). From 1927 to date, approximately 600,000 ft of diamond drill was put down. This work indicated that sufficient tonnage existed to justify a redesigning of the whole operation, surface and underground. After four years of general study, the following program was planned:

1. A new mine entrance, which would be an incline, that could follow the ore body down at whatever pitch was necessary. The incline will be equipped with conveyors for the moving of ore and waste to the surface and with tractor-type locomotives for man and supply transportation.
2. The new incline also required a new type of mining which was developed and is now in use. It is called contour mining and will be described in a future paper.
3. The new incline exit would necessitate the moving of the mill and mine shops across the Pend Oreille River. This part of the program is now underway.

The Incline

The sinking of the incline was to start as soon as World War II ended and was as follows:

The first leg of the incline was to be sunk from the surface 1600 ft on a 17° slope. The collar and first level at elevation 2180 ft, the second level at elevation 2000 ft, the third level at elevation 1875 ft, and the fourth level at elevation 1700 feet. From the 1700 ft elevation the incline was to flatten out to 12° for 400 ft to give the necessary depth for the ore pockets below the 1700 ft level and the necessary clearance for future sinking (see Fig 1 and 2).

Due to lack of manpower in 1946, the program was changed and was as follows:

A drift was driven from the old mine workings on the 1700 ft elevation in an easterly direction. At 1300 ft the drift was turned N 50° E and at this point a raise was driven 180 ft on a 50° slope. This raise intersected the Josephine horizon and commercial ore was encountered. At the 2000 ft mark, a main raise was driven, 245 ft on a 50° slope, and the 1875 ft elevation was cut. Exploration drifts were started on this level and production followed on a limited basis. The main drift at the 2500 ft point was turned N 35° E and ran parallel to and 10 ft east of and under the proposed incline line. At the proposed intersection of the drift and incline on the 1700 ft elevation, it was planned to raise the incline to intersect the 245 ft raise and to continue on to the surface, a distance of 1600 ft. When this proposed intersection point was reached, a heavy flow of water, approximately 800 gpm, was encountered and all work on the main drift face was stopped. This water flow flooded the main pump station in the old mine and the two lower levels with approximately 20,000,000 gal of water. The water was controlled and finally drained from the cave areas and lower levels after six months of pumping.

After the heavy flow of water was encountered in the main heading, it was decided that the incline would have to be started from the surface, as originally planned, so that too much time would not be lost. The surface overburden had to be removed, a total of 6000 yards. A temporary dry house for 6 men was built. An 8 in. churn drill hole was intersected in the first raise driven from the 1700 foot elevation tunnel. Air and water lines were placed in this hole, and air and water were delivered to the collar of the incline from the mine working. The incline started down at 15 ft wide and 7 ft high through the Leadbetter slates. After sinking 4 sets, it was

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TP 2608 A. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Jan. 31, 1950. Manuscript received Nov. 23, 1948.

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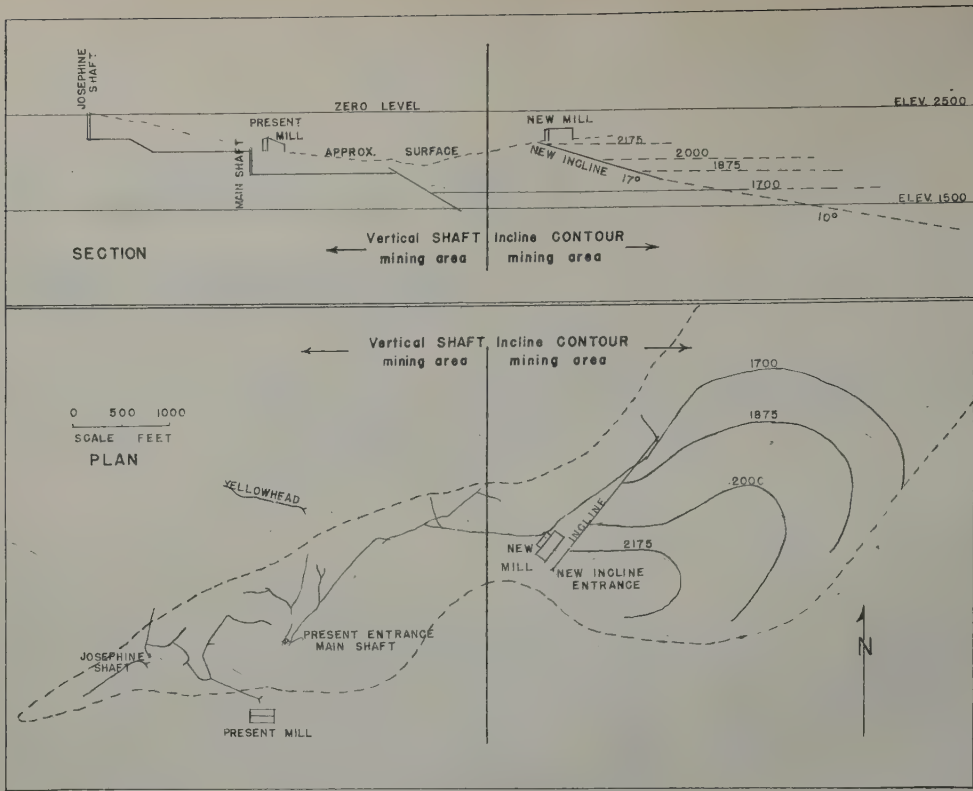

FIG 1 and 2—Plan of inclines to follow ore horizon.

decided that if the incline was 15 ft wide, it would have to be timbered a distance of 900 ft in the slates. It was then decided to drive the incline down 7 by 7½ ft, untimbered, and check the ground conditions.

The first section of this 17° inclined shaft, running from the surface to the 1875 ft level, has been recently completed. A distance of 1051 ft was sunk through shale and limestone in eight months time, using two slusher hoists with an average of four men per day, on a two shift basis.

This very fast sinking job was accomplished principally because of the rapid system for muck removal. Instead of using a hoist and skip in combination with a portable slide and scraper, as was originally planned, dragline slusher hoists were used throughout.

The shaft was started in August 1947, and was run 7 by 7½ ft, with rounds averaging 35 tons. The first 170 ft were sunk with three men mining and slushing on the same shift. A three drum, 15 hp Sullivan slusher hoist was set at the surface which cleaned the face in approximately 2 hr each morning, giving the miners a setup for the wagon drill (see Fig 3), and leaving the rest of the shift for mining and for dragging the muck the remainder of the distance to the surface. An RD-4 bulldozer removed the muck from the collar of the incline. From 170 to 276 ft, the same manner of sinking was used, except that a fourth man was added to the crew and work was done on two shifts. A slusher operator on day shift, cleaned the face while the muck was removed from the incline on night shift, at the same time that the drilling was being done.

On November 1st, at the 276 ft point the 15 hp, 3 drum (only 2 drums were used) slusher hoist was moved to a previously prepared station at the 200 ft point, while a 25 hp, 2 drum Sullivan slusher hoist was mounted on a 20 ft high platform on the surface. The platform was walled up on the incline side and two draw chutes installed. Thereafter all broken rock was removed from the incline collar by truck. From this point on, two slusher hoists were always used, the lower hoist cleaning the face of muck and the surface hoist pulling the muck to the incline collar. The 25 hp surface slusher hoist was replaced by a 50 hp double drum 325 ft per min Sullivan slusher hoist on December 13, 1947. The lower slusher hoist was moved ahead approximately every 200 ft to a station cut in the side of the incline.

FIG 3—Pend Oreille 100 lb wagon drill carriage.

No changes were made in slushing

procedure until February 7, 1948, at the 750 point, when mining was changed to the day shift and all slushing was done on the night shift. This change forced a slight overlapping of the scrapers, but this was done in front of the lower slusher hoist and in full view of the lower operator and no difficulty was ever experienced because of running the slushers simultaneously. A blinker system was installed in case either operator wished to stop the other.

All scrapers were of the hoe type and were built in the Pend Oreille shop. The 15 hp hoist used a 54 in. scraper with a ½ in. cable for the lead line and ⅜ in. for the haul back. The 50 hp hoist used 58 in. scraper, moving approximately ¾ of a ton. Two scrapers were used, placed from 15 to 300 ft apart for 867 ft of sinking and moving a load approximately 1½ tons. At this point the 50 hp motor burned out, due to overloading, so one scraper was cut off. Cable of ¾ in. diam was used on the 50 hp hoist lead line and cable of ½ in. on the haulback line. As only 600 ft of ¾ in. cable could be put on the drum of the 50 hp slusher, slushing had to be done in two steps. From 600 ft down the incline, muck was pulled up about 300 ft. This was done by putting in a portable cable, 200 ft long, which made a total length of 800 ft of cable between the hoe and the hoist. This portable cable was then removed, and the muck was pulled the additional 500 to 600 ft to the surface. The haulback line was held up to the back every 100 ft by a Sullivan 8 in. block, thus preventing any twisting of the lines (see Fig 4).

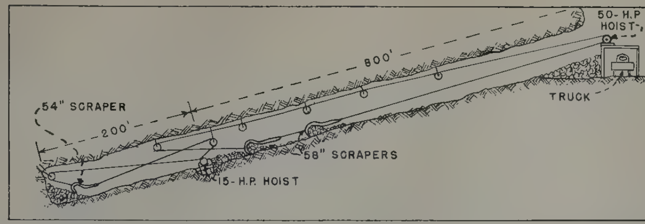

FIG 4—Slusher layout for scraping broken material in incline 1000 ft.

On April 17, 1948, with an average of 133 per month, or 1.3 ft per man shift, the shaft broke through to an underground level and this phase of the operation was completed.

After the slate section was completed, it was suggested by H. Anderson, Bulldozer Operator, that the bottom be removed and the incline cross section be 15 ft high and 7½ ft wide, so that the ground could be controlled without timber. Mr. Anderson's suggestion was followed and the conveyor belt will be in the upper half of the incline, the tractor in the lower half (see Fig 5). The double deck portion of the incline will be for the first 1400 ft and the incline will be 15 ft wide and 7 ft high (see Fig 6), for the next 1200 ft and for all future extensions. The incline is now being driven up from the 1700 level at 10° to meet the heading on the 1875 level.

Conveyor

The first section of the conveyor will be 1450 ft long with a 400 ft lift, the second section will be 1200 ft long with a 200 ft lift. The 1450 ft section will be equipped with Goodyear C-125 cord belt, with ⅜ in. cover, 30 in. wide. Types of conveyor idlers and drives for

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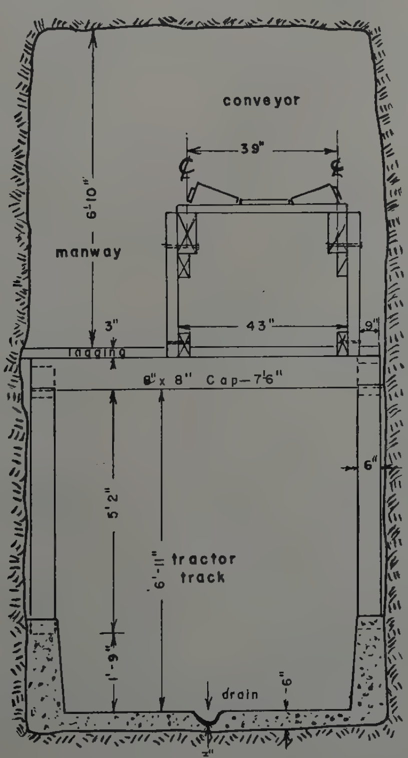

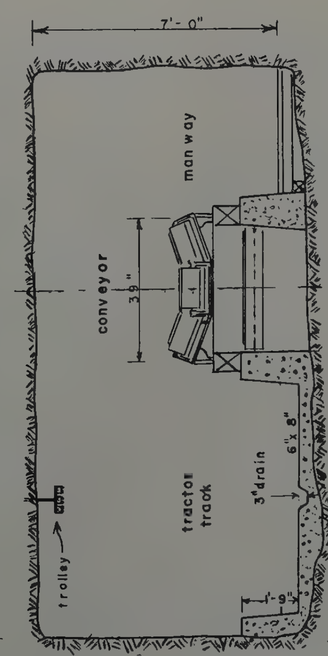

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FIG 5—Conveyor section of new incline.
FIG 6—Conveyor section of new incline.

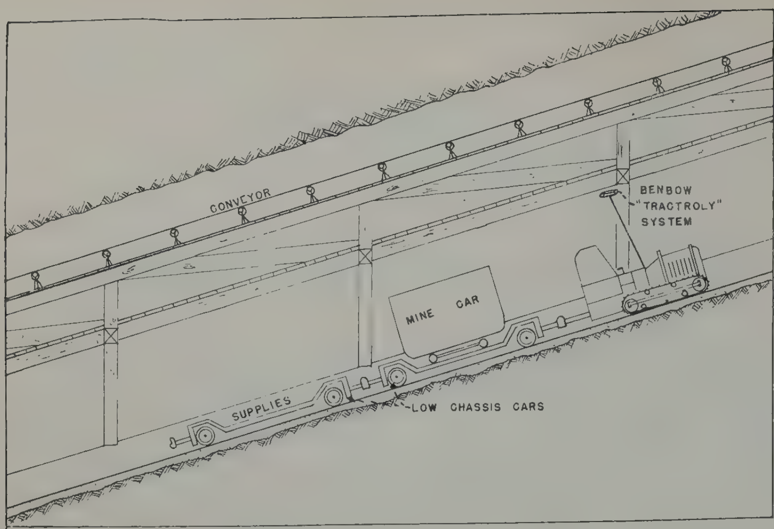

FIG 7—Electrified tractor locomotive and rubber-tired carry-all carts.

the incline are now being studied by the Engineering Department.

On each main ore loading level, there will be a crusher to reduce the mine run ore and waste to minus 2 in. material. The crusher discharge will be stored in a 1000 ton pocket above the loading point on the incline. Each loading point will be equipped with a loader which will control the feed to the conveyor at the required rate of 300 to 350 tph. Each loading point will also be equipped with a magnet to catch any tramp iron and steel that could injure the belt. The loading points will have push button electrical controls so that the belts can be started or stopped at any time.

Electrified Tractor Locomotive

A new method of transportation of men and supplies will be used in the incline. It will be an electrified, tractor-type mine locomotive to operate with an overhead trolley system. The electrified mine locomotive will be served by an overhead transmission line with a trolley "pickup." The electric service will be 440 volts, 60 cycle, three phase.

The Benbow "Tractroly" system, as manufactured by the Benbow Manufacturing Co., was selected as being particularly well adapted for the overhead transmission line. Each collector pickup has a positive multiple five tooth contact that eliminates arcing, flashovers, and reduces the possibility of the motor "single phasing." The conductor bars are enclosed in rubber insulation and offer no possible chance for human contact with a live conductor.

Power feeders to serve the overhead transmission line will be inserted at the required points to maintain a high operating voltage, with resulting low voltage drop. In this manner, the overhead trolley system can be extended as the incline is extended, without encountering high line losses or excessive voltage drop.

The electrified tractor is built from a standard farm type "Oliver Cletrac" tractor, with caterpillar type tracks. In order to electrify the tractor, the original four cylinder gasoline engine was removed. To support the electric motor, a new undercarriage was constructed and fastened to the tractor frame.

On the new undercarriage a 30 hp, 440 volt, 60 cycle, three phase, four speed motor has been mounted. A flexible coupling connects the electric motor to the tractor transmission, and the drive to the tracks is made through the normal method. The four speed motor has operating speeds of 1800, 1200, 900, and 600 rpm. These operating speeds result in tractor speed of approximately 5.25, 3.9, 2.6, and a slow speed of 1.3 mph.

The four speed electric motor is controlled by a manually operated four speed reversing drum controller. The drum controller is so constructed that the operator must select the operating speed before the electric motor is energized. A speed change can be made after the locomotive is in operation. A magnetically operated line switch is controlled by a pilot circuit on the drum controller, and provides undervoltage and overload protection for the electric drive motor. To complete the control equipment, an electric braking system is employed.

To improve traction, the tractor tracks are equipped with a special hard rubber track pad. These rubber pads are bolted to the tracks, which results in the tractor actually operating on a hard rubber tire.

The tractor locomotive will pull rubber-tired carry-all carts. These carts will be four wheeled, with under-slung axles. They will have air-type brakes that will set, if the carts become freed from the tractor locomotive. The cars will be equipped to handle men, timbers, steel, mine cars, and so on. There will be no rehandling of the loads. A mine car will be loaded on the surface, with a load for delivery to a given mine level. The operator will ride the tractor as a locomotive operator. This will reduce the signal system required, and the danger of giving wrong signals (see Fig 7).

Conclusion

In summing up the advantages of the incline, they are as follows:

1. The incline conveyor will deliver 350 tons per operating hour from the lowest level of the mine to the mill ore storage, a distance of 3500 ft with a crew of 2 men.
2. All ore and waste will be handled during one shift.
3. The tractor locomotive and carts will handle all men and supplies from the surface to the various mine levels, with one man per shift.
4. This incline can follow the Josephine horizon to its northern limit, 10,000 ft from the portal and 1300 ft below the portal.
5. The conveyor will move ore from the whole area and from the lowest levels at the lowest cost.
6. The tractor locomotive will service this incline, and all the working levels, and it is expected that transportation costs of men and supplies will be greatly reduced.

Acknowledgments

Mr. Lynn Kinney, Mine Superintendent, and Mr. Loren Billings, Engineer, and now Mine Foreman, assisted in the planning of this incline and the methods of transportation of ore and supplies. Mr. Hugh Tinling of Tinling and Powell Electrical Engineer's of Spokane, Washington, assisted with the planning of the tractor locomotive. Mr. J. H. Whitman prepared the sketches.

Aerial Magnetic Survey of the Vredefort Dome in the Union of South Africa

By OSCAR WEISS,* Member AIME

Summary

An aerial magnetometer survey was carried out by the author's geophysical organization over the Vredefort dome, where Witwatersrand beds are wrapped around a granite plug 25 to 30 miles in diameter. The well-known magnetic shales of the Lower Witwatersrand system (to be referred to as LWW) give strong negative anomalies over the out-cropping portion of the dome. This is unusual as these same shales cause positive anomalies over large areas near Johannesburg and along a strike of 130 miles between the East Rand and Klerksdorp.

It is suggested that the negative polarization is caused by heat and stresses connected with the doming.

This negative polarization of sedimentary beds, which usually cause positive anomalies, is a good example of how "magnetic lows" can be caused by "structural highs." Significant magnetic anomalies should be tested by gravity and seismic methods irrespective of the sign of the anomalies.

South of the line $X - Y$ (see Fig 2 and 5) the gravity anomalies are smaller over the LWW beds and magnetic anomalies suddenly become weaker and even change signs by becoming positive. This peculiar phenomenon is explained by faulting which truncated the nearly vertical LWW shales at depth. In this manner the mass of the LWW beds has been reduced together with the weakening of the negative magnetic anomalies by positive poles which occupy the truncated end. Furthermore, by uplifting the beds between f_1 and f_2 (Fig 2) positive polarization becomes predominant, thus causing positive magnetic anomalies.

General

The Vredefort dome is one of the internationally known geological structures, which, among others, has attracted the attention of numerous American geologists. It was for this latter reason that our aerial magnetic results over this structure were selected by us for publication in the United States of America.

A full description of the exposed geology has been published by Nel[1] and readers are referred to this paper, which is subject to only slight modifications inasmuch as the subdivision of the Witwatersrand system (to be referred to as WW system) is concerned.

The Vredefort dome is about 60 miles southwest of Johannesburg. The WW system, after being exposed in the vicinity of Johannesburg, plunges with southerly dips beneath the Ventersdorp and Transvaal systems. Along a section southwest from Johannesburg, as shown diagrammatically in Fig 4, we see no outcrops of the WW system until the Vredefort dome, where the Upper and Lower WW system are wrapped around an almost circular arc of a granite mass. Here the whole WW system is overturned around the perimeter of a huge granite plug 25 to 30 miles in diameter. Less than half of the circumference of this dome is exposed; the rest is covered by Karroo dolerites and sediments. The determination of the exact shape of the concealed part of the dome has been a matter of considerable interest and obviously geophysical methods were eminently suited for obtaining this information. A good start was made with a gravimeter survey carried out by the Geological Survey of the Union of South Africa.[2] Our own

San Francisco Meeting, February 1949.

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[1] References are at the end of the paper.

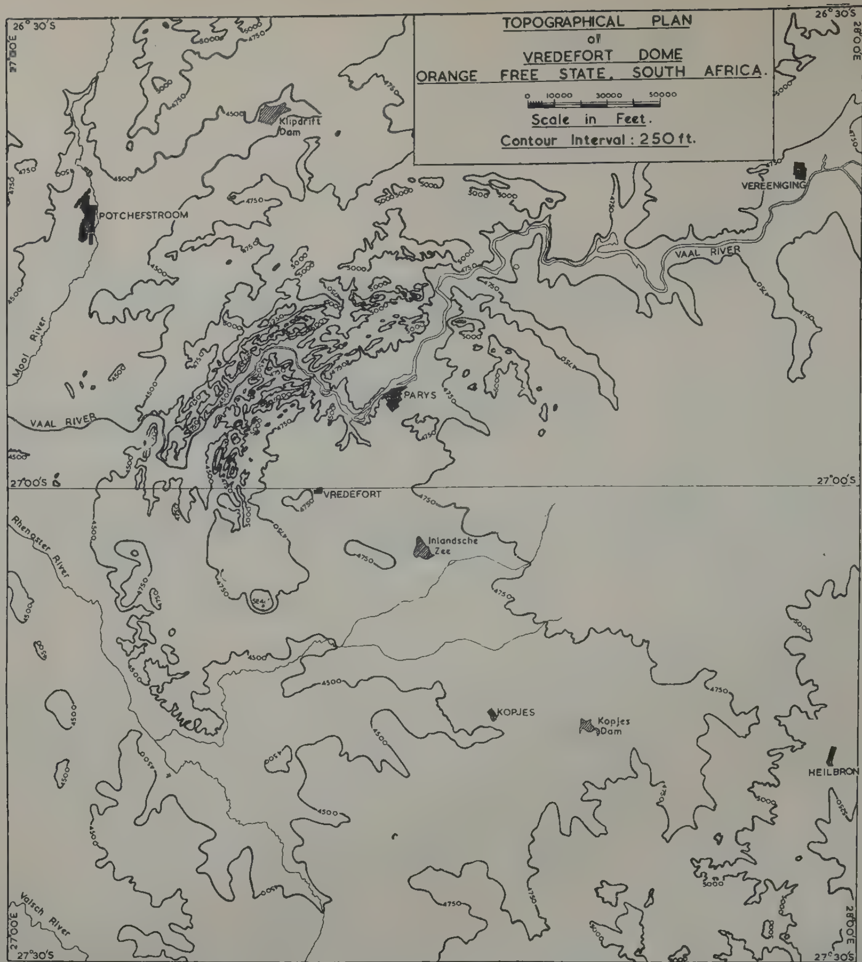

FIG 1—Topographical plan of Vredefort dome.

work was done during July 1948, and, incidentally, this was the first aerial magnetometer survey in South Africa, and perhaps in the whole of Africa.

The present survey was carried out by the author's own aerial magnetometer unit. The instruments were installed in a D.C. 3 (Dakota) aircraft, captained by D. S. Flett. He and his co-pilot, B. Cox, were responsible for the excellent navigation, which, combined with aerial strip photography, gave the position of the plane and hence the position of the magnetic anomalies. As fairly accurate surface plans of the area were available, we have not used our Shoran units for the aircraft's position-finding. The height of the plane was recorded by radio and barometric altimeters, but no correction was made for the sudden elevation changes of the ground, which can be seen from the topographic plan (Fig 1) of the area.

Results

Fig 2 shows the sharp negative anomaly over the outcrops. It is remarkable that these same magnetic shale horizons give positive anomalies in the East, Central, West, and Far West Rand over a strike of 130 miles with dips of about 25° to the south and southeast.

At this point we would like to mention that the sign of the anomalies was taken as positive if it increased the normal magnetic field, and as negative if it opposed the earth's field. By adopting this convention, anomalies in the Northern and Southern Hemispheres are directly comparable, and magnetically similar rocks under similar geological conditions show similar magnetic anomalies in both Hemispheres, although a positive anomaly in the Southern Hemisphere is caused by a north-seeking pole, while in the Northern Hemisphere a south-seeking pole will cause, in the above sense, a positive anomaly.

The negative anomalies over the outcropping LWW shales on the Vredefort dome are of the order of −1000 to −2000 gammas. Our observations at 1000 ft above ground over outcrops in the Central, West, and Far West Rand found anomalies ranging between +500 to +600 gammas over the LWW shales. From previous studies[3] we know the strongly magnetic shales and their vertical distance from the old granite in the West Rand. These are given in Table 1.

Table 1 . . . The Magnetic Shales and Their Vertical Distance from Granite, West Rand

Magnetic Shales	Vertical Distance above Granite, Ft
Water Tower slates	800
Contorted beds	2,500
Lower Hospital Hill shales	5,000
Middle Hospital Hill shales	6,700
Upper Hospital Hill shales	8,800
Promise beds	10,000
West Rand shales	14,000

Observations on the ground disclosed that vertical magnetic anomalies over outcrops of the weaker magnetic shales were positive and of the order of 1000 to 1500 gammas, but the Water Tower slates, contorted beds, and West Rand shales displayed alternating negative and positive polarization ranging from −5000 to +5000

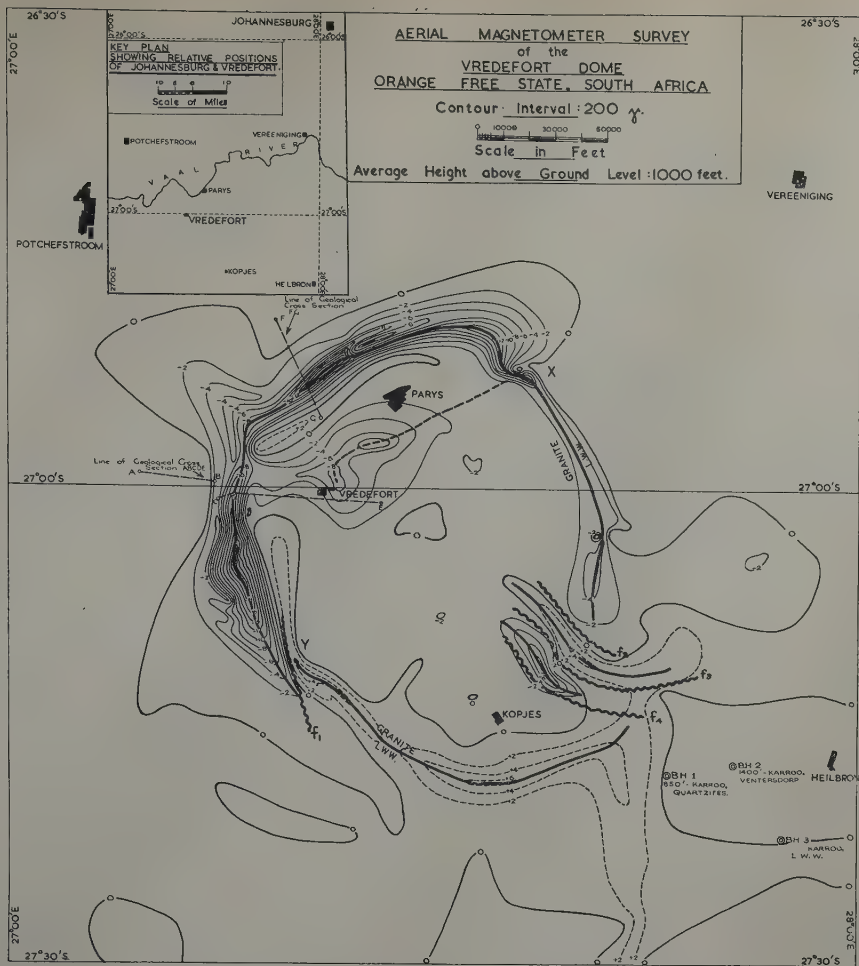

FIG 2—Aerial magnetometer survey of Vredefort dome.

gammas on outcrops. Under the cover of 1000 ft thick dolomite, the Water Tower slates and contorted beds gave positive anomalies of about 650 gammas, while under the cover of 3000 ft thick dolomite the anomaly of the West Rand shales was positive and of the order of 300 gammas. On the other hand, on the East Rand the Water Tower slates and contorted beds gave positive and negative vertical anomalies, the strongest of them varying from −10,000 to +10,000 gammas at places where the cover must have been only a few hundred feet thick.[4] We can state that in the numerous and widespread observations made on the ground by vertical magnetometers over LWW shales, the anomalies were always positive with the exception of the Water Tower slates and contorted beds in the Far East Rand, which had strong positive and negative anomalies.

Fig 2 shows the magnetic anomalies of the total magnetic intensity as recorded from the average height of 1000 ft above the ground. Over the hilly portion of the topography the height of the plane may have been as much as 250 ft lower over suddenly rising hills formed by outcrops of the LWW system. As the intensity of the magnetic anomalies varied along the strike of the beds even over flat terrain, it would have been difficult to attempt magnetic corrections for the height of the plane. Another difficulty in applying corrections is due to the fact that the magnetic effect measured in the aeroplane is the integral of a magnetic volume, the depth of which is unknown. This problem can be solved by recording the magnetic anomalies at two or three different heights.

Returning to Fig 2, a narrow belt of sharp negative magnetic anomalies follows the outcrop of the LWW shales. The axis of this anomaly is close to the base of the LWW system, thus confirming the geologically mapped dips which show the overturning of the WW system. As the axis of the anomalies does not extend far inside the area covered by old granite, we believe that the flatter angles of the LWW shales, i.e., dips of 50 to 70°, do not continue to great depth. They may be caused by near-surface slumping of the overturned beds, which at depth are nearly vertical.

The intensity of the anomalies of the total magnetic vector over the outcropping LWW shales varies between −1000 and −2000 gammas. The width of the belt of anomalies varies from 13,000 to 28,000 ft, increasing with the intensity of the anomalies.

In the northeast corner of Fig 3 at point X where the Karroo sediments and dolerite sills begin to cover the LWW shales, anomalies suddenly become weaker, and are of the order of −200 to −400 gammas, but they remain negative up to fault f_2.

At point Y in the southwest corner of Fig 3, the LWW shales disappear beneath Karroo sediments. At this place the magnetic anomalies suddenly become positive of the order of +400 to +600 gammas, and this positive anomaly continues to fault f_4. The anomaly picture suggests fault f_1 near point Y.

Between faults f_2 and f_3 positive anomaly coincides with a block of

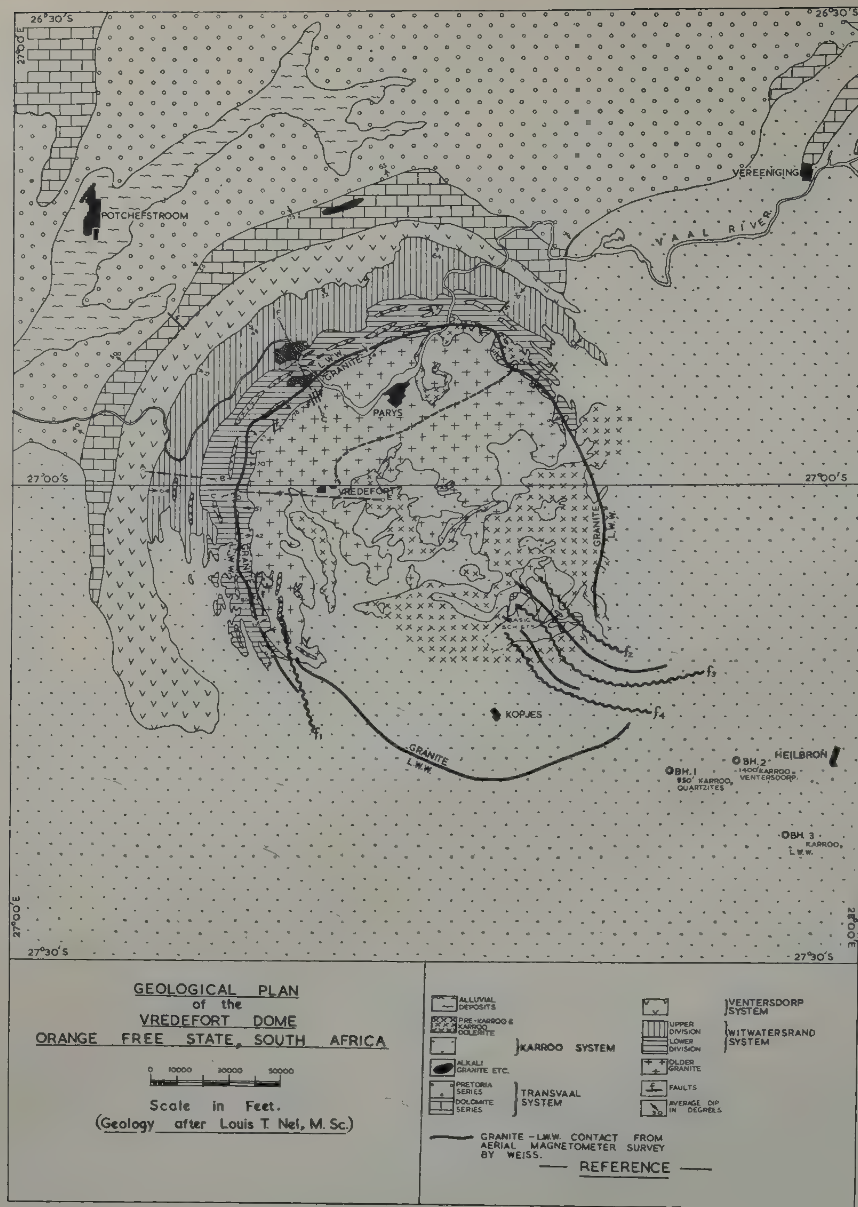

FIG 3—Geological plan of Vredefort dome.

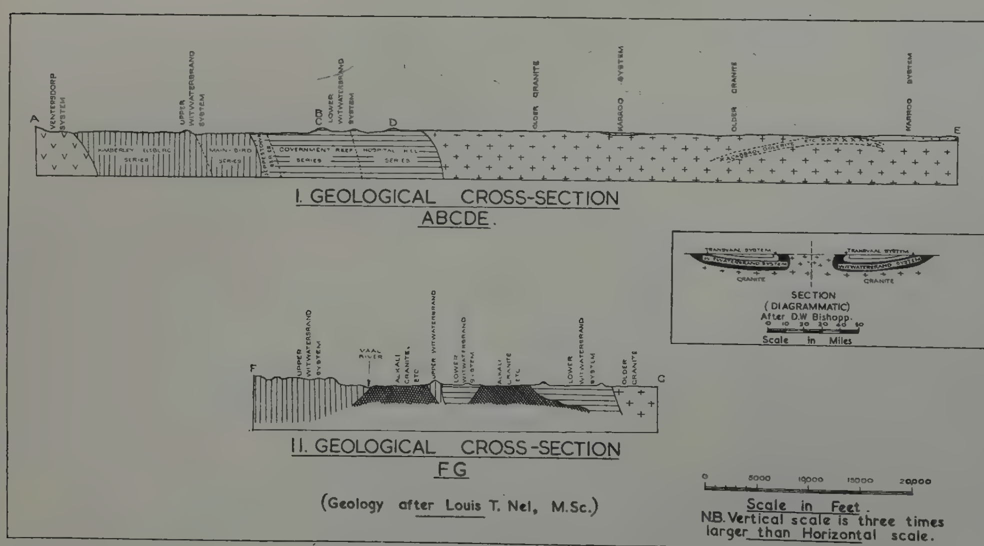

FIG 4—Detail of sections of Fig 3.

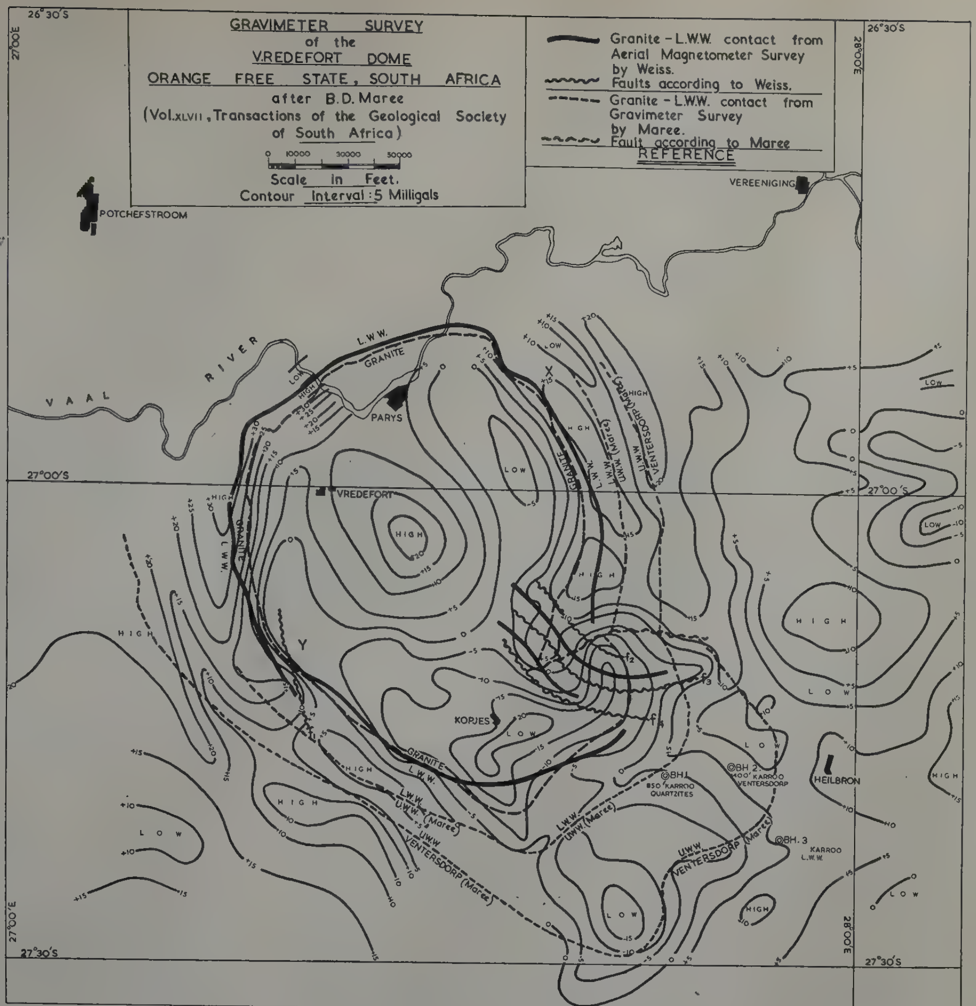

FIG 5—Gravimeter survey of Vredefort dome.

basement schists, which has probably been preserved between these two faults. All the above sub-Karroo faults are postulated from the magnetic results.

Between faults f_3 and f_4 a strongly negative anomaly is noticeable. This may be the block of LWW shales which originally bridged the gap between faults f_2 and f_4.

We have to estimate the thickness of the Karroo beds between points A and B. Borehole No. 1 west of Heilbron intersected 850 ft of Karroo beds. Their thickness over the LWW system between A and B is likely less and is of the probable order of 300 ft.

Some weakening of the negative anomalies at X is understandable as the Karroo cover blankets the magnetic rocks. On the other hand, the sudden change of the anomalies from negative to positive at Y is a very surprising feature. This change is not caused by any important changes in the strike and dip of the LWW beds. The strike of the positive zone in the south is the same as that of the negative zone in the north. The dips remain nearly vertical, as can be seen from the nearly symmetrical shape of the anomalies. A slight dip away from the vertical and away from the granite is suggested in the magnetic contours. The area of positive anomalies is distinguished by lack of Karroo sills on the surface.

Nel's original geological map shows that the LWW system is shot through and through with sills of Karroo dolerites, as well as by pre-Karroo sills of gabbros, epidiorites, etc. In our earlier remarks, we emphasized that the strong negative anomalies of the LWW shales at the Vredefort dome were exceptional. Over 130 miles of strike and more near the Rand, the LWW shales give positive anomalies. The only exception was the Water Tower slates and contorted beds on the Far East Rand.

We could first suggest that the negative anomalies at Vredefort are the results of polarization through the heat of the sills. If this were true, then we should not have sills in and over the LWW shales in the zone of positive magnetic anomalies. As we have no boreholes in this area, we have to use geophysical information. Gravimeter measurements should show lower values over the LWW shale horizon within the positive magnetic zone. An inspection of Fig 5, which is Maree's[2] gravimeter plan confirms our opinion. It shows that while in the negative magnetic zone the gravity anomalies are between +15 and +25 milligals over the LWW system, in the positive zone the gravity anomalies over the LWW shales are +5 milligals and less. In fact, the positive anomaly disappears south of Koppies. The absence of gravity anomalies over the LWW shales is difficult to explain.

We could assume that the sudden decrease of the gravity anomalies was due to lower density differences between the granite and the LWW shales. This is very improbable as the densities of these rocks are well known and the

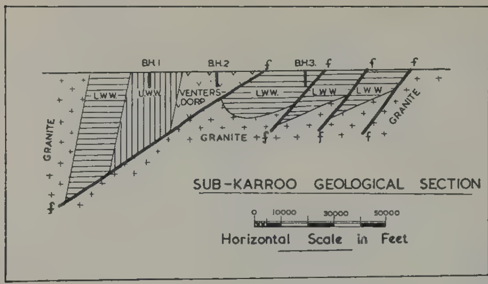

FIG 6—Sub-Karroo geological section.

LWW shales are about 0.16 heavier than the granite. South of X, where the strong negative anomalies of the outcrops of LWW shales drop from −1200 to −400 gammas, we have smaller *negative* anomalies but the general level of magnetic intensity increased. Further south, south of f_4, the general level again increased and the anomalies themselves became positive, so that we have lower gravity but higher general magnetic intensities.

We could explain the decrease of the magnetic anomalies by simply assuming greater thickness of cover, but we know that the cover is only about 300 ft over the LWW shales south of X, and we know from our own aerial magnetometer observations that a height of 2500 ft is required above outcrops of LWW shales at Vredefort to reduce the anomalies from about −1000 gammas at 1000 ft height to about −400 gammas at 2500 ft height.

It is interesting to note that over the positive anomalies south of Koppies the increase in height from 1000 to 2000 ft decreased the anomaly from +400 to +200 gammas, but another thousand feet increase in height, from 2000 to 3000 ft, caused only about 50 gammas decrease.

When we realize the lower anomalies and higher general level of the magnetic intensities south of X and Y, together with the lower gravity anomalies of this same area, we have to consider whether the lower gravity anomaly is not due to a reduction in the mass of the LWW system and whether the lower negative anomaly is not due to the same cause coupled with the increased effectiveness of positive magnetic poles which balance the negative effects. This leads us to the idea of faults cutting off the LWW system at depth, thus reducing mass and bringing positive magnetic poles to the truncated lower end of the LWW shales (as shown by Fig 6). The abnormal negative anomalies are proofs of polarized magnetism, and we mentioned repeatedly that normally over large areas and over great length of strike the LWW shales give positive anomalies. If the negative polarization is confined to the upper few thousand feet of the LWW shales, then positive magnetic polarization will exist at increasing depths. If we now truncate the nearly vertical limb of the LWW shales then the negative polarization will be first reduced. If in addition the block between f_1 and f_2 has been vertically upthrown, the positive polarization could be brought nearer the surface and become predominant. We know that the LWW shales possess polarized and not induced magnetism. Thus the shales cannot be considered as a single magnet. The positive and negative polarities are distributed according to the magnetite concentration in the shales and according to the effectiveness of the causes of polarization. At the same time the gravity anomalies will be weaker over the truncated LWW beds. The latter condition can be observed from Maree's results. Although Maree shows it, yet in fact it is not possible to locate the various contacts from gravity data in the southern quadrant of the dome, and in this same quadrant the predictions of Maree's interpretation were contradicted by boreholes, as Ventersdorp lavas were in hole No. 2 and LWW beds in hole No. 3, although his plan claimed Upper WW in borehole No. 2 and Ventersdorp in borehole No. 3.

With the adoption of the idea of truncation and uplift of LWW beds by faulting in the area south of X–Y, we do not need to postulate the heat of intrusions as the cause of negative polarization. The cause can be sought in the heat and stresses accompanying the doming.

The remarkable polarization at the Vredefort dome is a good example of what can happen in unknown areas covered by younger beds. A dome can exist beneath a "magnetic low," i.e., beneath a negative anomaly. If the Vredefort dome were covered by younger beds we would have a magnetic low over the dome, which would very probably be mistakenly interpreted as a structural low. The lesson is that every significant magnetic anomaly should be tested by gravity and seismic methods, quite independently of the positive or negative nature of the anomalies.

Acknowledgments

Thanks are due to the staff of my aerial magnetometer unit; to Mr. H. Fletcher, Physicist in charge; Mr. A. T. Roux, Party Chief of the first magnetometer unit; Mr. Bradford, Chief of plotting, and to Mrs. Krsic, who drew the diagrams and plans for this paper.

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Performance Tests of an Experimental Installation of Cyclone Thickeners at the Shamrock Mine

By T. FRASER,* R. L. SUTHERLAND,† Members AIME, and F. F. GIESE‡

Under a cooperative agreement between United States Bureau of Mines and the Truax-Traer Coal Company, some operating-scale experiments have been made with the cyclone thickener in the preparation plant of the Shamrock No. 1 mine at Kayford, West Virginia. These tests were made with a 14-in. cyclone of the type developed by the Dutch State Mines in Limburg. It was applied experimentally to the thickening of solids in washery water to obtain performance data to show the range of sizes that may be recovered and other operating characteristics that would indicate the field of usefulness of this device in coal preparation practice.

This is a progress report only. It covers results obtained in a series of 17 tests of the operation of what has been considered a unit of commercial size operating on suspensions of fine coal mixed with a relatively small amount of free impurities. The underflow material, after further dewatering in centrifuges, is a marketable grade of coal.

The coal at Shamrock No. 1 is mined from the No. 2 Gas seam. It is made up essentially of a low ash coal suitable for byproduct use with one or more bands of splint coal of higher ash content suitable for industrial fuel.

The washing plant was installed for the two-fold purpose of removing the free impurities incident to mechanized mining and to separate the run of mine coal into the two types. To accomplish these results it is necessary to make a separation at a specific gravity of about 1.4 to recover the byproduct coal and to separate the splint coal from refuse at a higher gravity.

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The coarse coal section of the washing plant was put into service about 10 months before the fine coal section was completed. Inadequate screening capacity in the raw coal preparation plant resulted in a portion of the fine coal being fed to the jigs. No provision for preliminary dedusting of the raw coal feed was incorporated in the design.

Under the conditions in the initial operating period, separation at the desired lower gravity was not realized, apparently because of the presence of an excessive amount of fine solids in suspension in the wash water. Difficulty in wetting the fine coal may also have been a contributing factor.

Additional facilities for the removal of the fine solids from the wash water was indicated. Information then available on the Driessen cone for this purpose was studied and a cooperative agreement with the Bureau of Mines for experimental work with a unit of standard size as developed by the Dutch State Mines was arranged.

During the time required for construction of the cone it was found the addition of a small amount of wetting

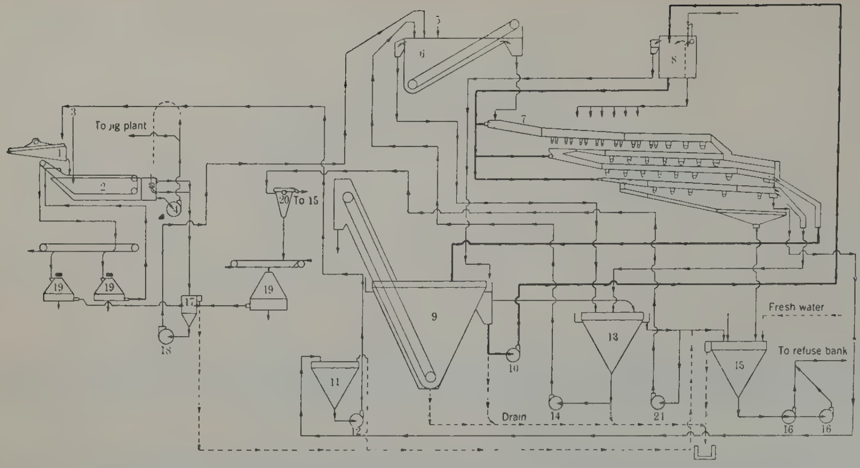

FIG 1—Flowsheet of the fine coal system of the washery at Shamrock mine No. 1

agent to the wash water brought about a better settling action and made separation at the desired lower gravity possible. However, a considerable amount of fine material remained in suspension, some of which was lost in the overflow from the settling tank.

While the original interest of the company in the possible value of cones in the washing plant was in a reduction in the suspended solids in the wash water to improve the effectiveness of washing, two other conditions affecting cost of operation were considered. These were (1) the recovery of marketable coal from the slurry pumped to the refuse pond, and (2) a possible reduction in makeup water required.

Recovery of Solids

Examination of the solids pumped from the refuse tanks indicated it was made up of (1) rejects from the Rheo plant containing negligible amounts of float material or material below 48 mesh which is discharged directly to the refuse sump, and (2) fine coal of marketable grade from overflows from other parts of the circuit. Measurement of the amount of solids pumped to waste indicated that approximately 3 pct of the total run of mine product above 200 mesh was being lost in the refuse. Of this amount, approximately half was coarser than 48 mesh and half between 48 and 200 mesh.

Water Supply

The supply of readily available water is limited. All available water is acid and requires treatment before use in the plant. The cost of providing, pumping, and treating the volume of water now required adds substantially to the operating cost. In addition, all dirty water now discharged with the refuse is filtered through a bed of coarse refuse before re-use or discharge to the drainage stream. Removal of the fouled filter material and its replacement is an added operating cost.

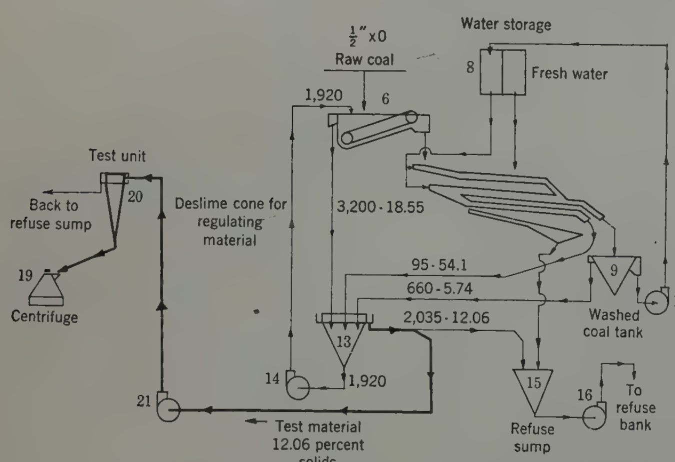

FIG 2—Diagram of experimental installation showing connecting lines and test material sources.

In order to reclaim water in this plant, it is necessary to dewater the rejects from the Rheo plant separately to permit disposal with the dewatered rejects from the jigs. This would leave only the overflows from the several settling cones to be treated for removal of marketable coal, with at least a portion of the partly clarified water returned to the plant circuit.

The usefulness of the cones in this operation will be determined by the tonnage of marketable coal recovered and the amount of water that can be returned to the circuit.

The Shamrock plant is a combination jig and Rheolaveur washery of a rated capacity of 400 tons per hour. The 2 multiple-compartment Baum-type jigs handle the 4 by ½ in. coal and a 5-trough Rheolaveur free-discharge washery handles the ½ in. by 0 coal. The details of the water- and sludge-handling system and other elements of the plant that are pertinent to the understanding of the water-clarification problem are shown in the flowsheet, Fig 1.

Water-circulating System

The coarse-coal washery and the fine-coal washery have substantially independent water systems, connected only by minor diversion of overflow water from the jig-washery settling tank to the Rheolaveur feed and a roughly equivalent return of wash water with the secondary steam coal product which drains back into the same jig-washery settling tank.

The experimental cyclone thickener was installed in the fine coal washery circuit. In this plant the washing unit is a 5-launder Rheolaveur, fed by a drag-conveyor-discharge tank that serves to wet the feed coal before it goes into the washery. This raw-coal wetting tank is item No. 6 in the flowsheet. The water supply for this tank is drawn from the general water-circulating system; and its overflow, carrying some fine raw coal, returns to the Rheolaveur circulation, adding to the complexity of the water-clarification problem.

Table 1 . . . Size Consist of Typical Sample of Test Line Solids

Size Range, Mesh	Weight, Per Cent	Ash, Per Cent[a]	Cumulative, Per Cent	
			Weight	Ash[a]
Plus 4	0.00		0.00	
4 × 6	0.03		0.03	
6 × 8	0.04		0.07	
8 × 10	0.13		0.20	
10 × 14	0.36		0.56	
14 × 20	0.96		1.52	
20 × 28	3.14		4.66	
28 × 35	2.76		7.42	
35 × 48	14.84		22.26	
48 × 60	6.02		28.28	
60 × 80	9.16		37.44	
80 × 100	4.12		41.56	
100 × 150	10.62		52.18	
150 × 200	7.29	5.57	59.47	5.57
200 × 250	2.30		61.77	
250 × 270	1.80		63.57	
270 × 325	2.66	10.00	66.23	6.02
325 × 0	33.77	18.23	100.00	10.14

[a] Ash tests were made only on composite samples of +200 mesh; 200 × 325 mesh and −325 mesh. Thus in column 3, 5.57 pct ash is the analysis of a composite of all the size fractions shown, down to and including 150 × 200 mesh.

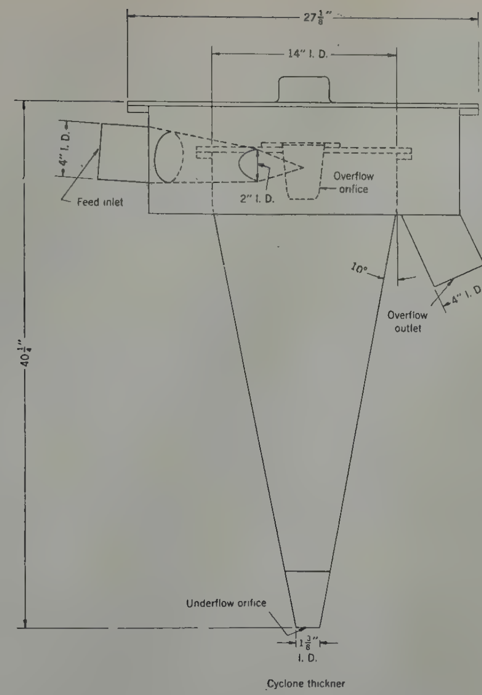

FIG 3—The experimental cyclone thickener.

The main water circuit in the Rheolaveur plant, indicated by the heavy line in Fig 1, is relatively simple and conventional. Push water for the launders is drawn from the dirty water compartment of head tank 8, and hydraulic water is drawn from the fresh water compartment. The wash water that goes with the washed coal from the top launders reaches the washed coal draining cone 9, and from there it overflows to pump 10 and is returned to the head tank.

The secondary circuit, handling the circulation of regulating material in the Rheolaveur plant, consists primarily of collecting and desliming cone 13 and circulating pump 14. Middlings from the *C* launder, which constitutes the regulations fraction, goes directly to the desliming cone. This cone also receives the overflow water from raw coal wetting tank 6 and excess water from washed coal dewatering cone 9; both of these lines carry in some solids, mostly clean coal. The settled solids, with enough water for pumping is returned to raw coal wetting tank 6, where it joins the feed back to the Rheolaveur washery, thus completing the regulating return circuit.

Effluent water from regulations desliming cone 13 goes to refuse pump sump 15, where it joins the Rheolaveur washery rejects and is pumped to the refuse bank.

A part of this effluent water from regulations desliming cone 13 constituted the test material for experiments with the cyclone thickener. Fig 2 shows the connections made to take off the test water from the desliming-cone effluent line and the immediate connections with the Rheolaveur water system that supply the material. The heavy line shows the new work installed to make up the test unit line. It takes dirty water from the regulations-desliming-cone effluent line and delivers it by pump 21 to 14 in. cyclone thickener 20. The thickened sludge is

Table 2 . . . Size Tests of Source Materials That Go into Regulation Desliming Cone

Size	Wetting Tank Effluent, Per Cent				Rheolaveur Middlings, Per Cent				Clean Coal Effluent, Per Cent				Composite, Per Cent			
	Weight	Ash[a]	Cumulative		Weight	Ash[a]	Cumulative		Weight	Ash[a]	Cumulative		Weight	Ash[a]	Cumulative	
			Weight	Ash[a]			Weight	Ash[a]			Weight	Ash[a]			Weight	Ash[a]
½ × ⅜	0.83		0.83		0.99		0.99						0.80		0.80	
⅜ × 5/16	1.61		2.44		2.16		3.15						1.56		2.36	
5/16 × 4 mesh	1.50		3.94		7.88		11.03						1.94		4.30	
4 × 6	1.71		5.65		10.81		21.84		0.06		0.06		2.37		6.67	
6 × 8	1.50		7.15		10.13		31.97		0.12		0.18		2.13		8.80	
8 × 10	1.15		8.30		11.36		43.33		0.09		0.27		1.94		10.74	
10 × 14	2.60		10.90		9.25		52.58		0.18		0.45		3.01		13.75	
14 × 20	7.92		18.82		6.46		59.04		0.24		0.69		7.40		21.15	
20 × 28	12.64		31.46		6.48		65.52		0.44		1.13		11.50		32.65	
28 × 32	6.95		38.41		2.64		68.16		0.33		1.46		6.26		38.91	
32 × 48	20.39		58.80		5.81		74.00		2.37		3.83		18.26		57.17	
48 × 60	5.79		64.59		1.46		75.46		2.10		5.93		5.25		62.42	
60 × 80	8.97		73.56		2.14		77.60		4.33		10.26		8.18		70.60	
80 × 100	1.54	9.65	75.10	9.65	0.76	10.75	78.36	10.75	6.73		16.99		1.75	9.58	72.35	9.58
100 × 150	6.04		81.14		1.80		80.16		10.88		27.87		5.96		78.31	
150 × 200	3.73		84.87		1.13		81.29		8.60		36.47		3.77		82.08	
200 × 250	0.94		85.81		0.28		81.57		2.58		39.05		0.96		83.04	
250 × 270	0.68		86.49		0.20		81.77		2.22		41.27		0.73		83.77	
270 × 325	1.99	16.05	88.48	10.62	3.63	12.73	85.40	10.91	11.91	6.84	53.18	6.48	2.64	15.28	86.41	10.51
325 × 0	11.52	25.35	100.00	12.31	14.60	21.85	100.00	12.51	46.82	22.81	100.00	14.13	13.59	24.93	100.00	12.47

[a] Ash tests were made only on composite samples of +100 mesh; 100 × 325 mesh and −325 mesh. Thus in column 3, 9.65 pct ash is the analysis of a composite of all the size fractions shown, down to and including 80 × 100 mesh.

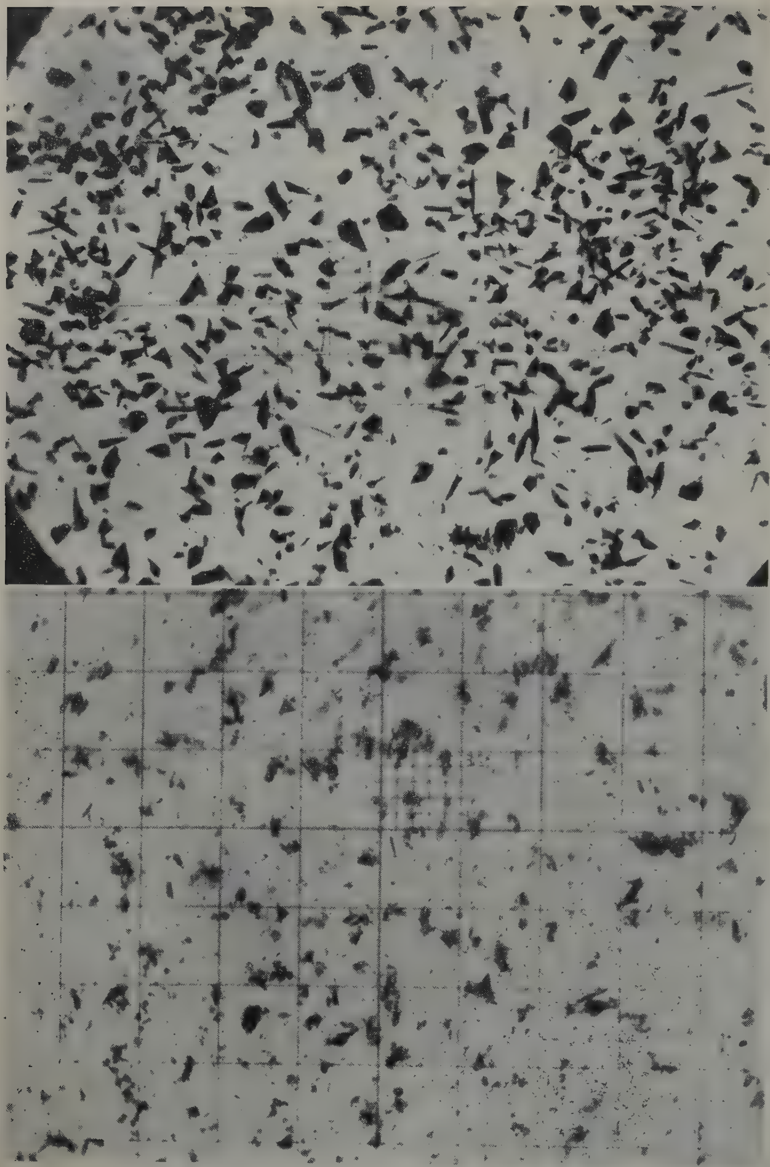

FIG 4—Micrographs of 20 by 10 micron particles.

delivered to the fine washed coal conveyor to be fed to centrifugal driers 19; the final effluent water from the experimental unit goes back to join the main plant water circuit again at refuse pump sump 15.

The numerical data on the flow lines of Fig 2 show the rate of flow in gallons per minute and the percentage of solids. For example, the effluent from the regulations desliming cone measured 2035 gpm and carried 12.06 pct solids by weight. This is the cyclone test material. There was some variation in actual percentage of solids during the test period, and these fluctuations are revealed in the record of test data.

Source of Test Material

Table 1 gives a typical size analysis of the solids in the effluent from the regulations desliming cone used as the test material. To further identify the sources of this material, data on the intermediate products that pour into this cone are given in Table 2. The largest source of fines that go into the test line is the overflow water from Item 6, the raw coal wetting tank. This water carries 18.55 pct solids of 12.3 pct ash content. The coarser material settles out in the regulating material tank and is returned to the Rheo circuit.

Effluent water from the clean coal dewatering tank, item 9, also carries coal into the test system, but this constitutes a relatively small proportion of the total. The data in the last column of Table 2 showing the total composite

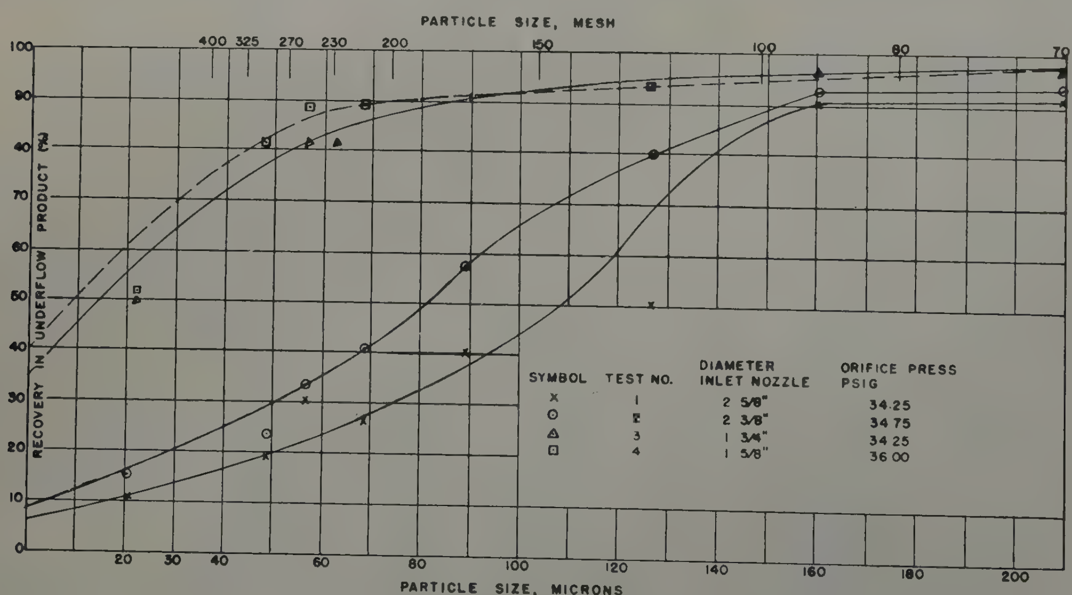

FIG 5—Results of Tests 1 to 4.

feed to the cone, are sample test data—not a statistical composite of the itemized data of the preceding columns.

Test Unit

The cyclone thickener used in the test is a 14 in. unit with 20° cone built substantially in accordance with the design of the Netherlands State Mines in Limburg.* Fig 3 shows the general design features. The cyclone is pumped by a 6 by 4 in. centrifugal pump driven by a 25 hp motor. The line loss between the pump and the cyclone inlet nozzle is 15 to 17 lb, with the pump running at 1168 rpm. Pumping water, the motor takes a load of 9.5 kw at that speed.

Test Procedure

The thickener tests were made during routine operation of the preparation plant from October 5 to 13, 1948.

The rate of flow through the unit and the inlet nozzle pressures were set for each test by adjusting the pump speed. The ratio of overflow orifice diameter to underflow was set in each instance by the diameter of the overflow nozzle, which is removable and interchangeable. The underflow spigot has a fixed diameter of 1⅜ in., and the inlet feed nozzle is fixed at 2 in. id.

The solids in the test material were subject to varying operating conditions in the washery and were not con-

* M. G. Driessen: The Use of Hydraulic Cyclone as Thickeners and Washers in Modern Coal Preparation. *Coal Tech.*, TP 2135 (Aug. 1947); *Trans.* AIME (1948) 177, 240.

Table 3 . . . Product Size Data of Cyclone Test 1

Size, Mesh	Underflow Product[a]			Overflow Product[b]		
	Grams	Per Cent	Cumulative, Per Cent	Grams	Per Cent	Cumulative, Per Cent
4 × 6	0.2	0.01	0.01	0.0	0.00	0.00
6 × 8	0.4	0.02	0.03	0.0	0.00	0.00
8 × 10	1.8	0.11	0.14	0.0	0.00	0.00
10 × 14	9.5	0.57	0.71	0.0	0.00	0.00
14 × 20	26.4	1.59	2.30	0.2	0.06	0.06
20 × 28	99.0	5.94	8.24	0.2	0.06	0.12
28 × 32	87.0	5.22	13.46	0.2	0.06	0.18
32 × 48	432.0	25.93	39.39	1.6	0.46	0.64
48 × 60	176.0	10.57	49.96	1.5	0.43	1.07
60 × 80	255.0	15.32	65.28	5.4	1.56	2.63
80 × 100	179.0	10.75	76.03	4.3	1.24	3.87
100 × 150	131.0	7.86	83.89	31.3	9.04	12.91
150 × 200	84.0	5.04	88.93	29.9	8.63	21.54
200 × 250	23.4	1.40	90.33	15.4	4.45	25.99
250 × 270	15.0	0.90	91.23	8.1	2.34	28.33
270 × 325	15.1	0.91	92.14	16.0	4.62	32.95
325 × 0	131.0	7.86	100.00	232.0	67.05	100.00

[a] Underflow rate, 55 gal in 1.77 min. Rate of flow, 31.0 gpm.
Sample weight (solids + water), 3370 g; dry solids, 1823 g; solids per cent, 54.1
[b] Overflow rate, 106 gal in 0.48 min. Rate of flow, 221.0 gpm.
Sample weight (solids + water), 4770 g; dry solids, 346.8 g; solids per cent, 7.27.

Table 4 . . . Work Sheet Showing Computations of Percentage of Recovery Data of Test 1

(Yield of underflow product 54.3 pct; yield of overflow product 45.7 pct)

(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)
Size, Mesh	Per Cent in Under-flow	Per Cent in Over-flow	(2) × Ratio[a]	(3) + (4)	Recovery (4) ÷ (5)	(2) × Yield Under-flow	(3) × Yield Over-flow	Cal. Feed (7) + (8)
4 × 6	0.01				100.00	0.01	0.00	0.01
6 × 8	0.02				100.00	0.01	0.00	0.01
8 × 10	0.11				100.00	0.06	0.00	0.06
10 × 14	0.57				100.00	0.31	0.00	0.31
14 × 20	1.59	0.06	1.89	1.95	96.92	0.86	0.03	0.89
20 × 28	5.94	0.06	7.07	7.13	99.16	3.23	0.03	3.26
28 × 32	5.22	0.06	6.21	6.27	99.04	2.83	0.03	2.86
32 × 48	25.93	0.46	30.86	31.32	98.53	14.08	0.21	14.29
48 × 60	10.57	0.43	12.58	13.01	96.69	5.74	0.20	5.94
60 × 80	15.32	1.56	18.23	19.79	92.12	8.32	0.71	9.03
80 × 100	10.75	1.24	12.79	14.03	91.16	5.84	0.57	6.41
100 × 150	7.86	9.04	9.35	18.39	50.84	4.27	4.13	8.40
150 × 200	5.04	8.63	6.00	14.63	41.01	2.74	3.94	6.68
200 × 250	1.40	4.45	1.67	6.12	27.29	0.76	2.03	2.79
250 × 270	0.90	2.34	1.07	3.41	31.38	0.49	1.07	1.56
270 × 325	0.91	4.62	1.08	5.70	18.95	0.49	2.11	2.60
325 × 0	7.86	67.05	9.35	76.40	12.24	4.27	30.64	34.91

[a] $\text{Ratio} = \dfrac{\text{Yield of Underflow Product}}{\text{Yield of Overflow Product}} = \dfrac{54.3 \text{ pct}}{45.7 \text{ pct}} = 1.19 \text{ pct}$

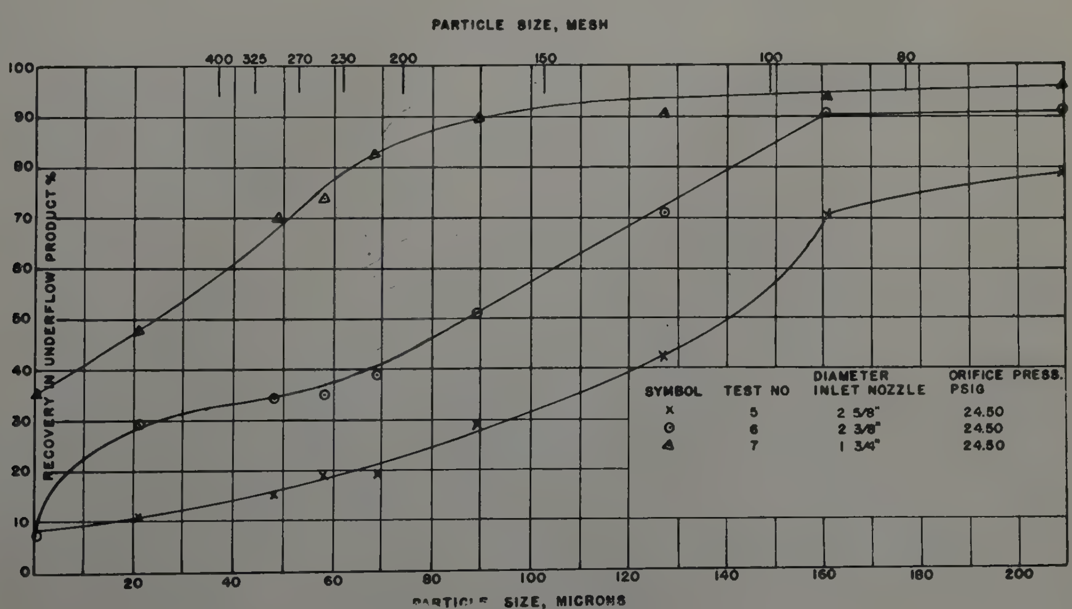

FIG 6—Results of Tests 5 to 7.

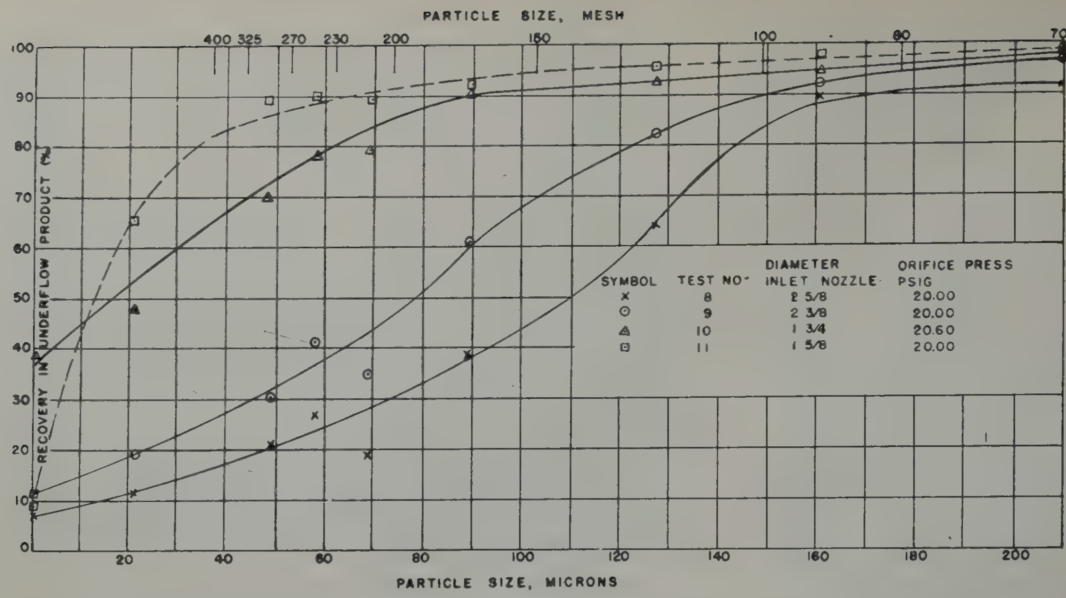

FIG 7—Results of Tests 8 to 11.

trollable by the operator of the experimental unit, but the proportions of solids and the size consist of solids were determined for each test.

The test data were obtained by the following procedures:

1. *Flow rate:* The rate of flow, in gallons per minute, was measured both at the overflow and at the underflow spigot by observing the time required to fill a 55 gal steel drum.

2. *Sampling:* The effluent product and the underflow product were sampled by passing a water sampler across the entire stream where it falls freely from the discharge line in each case. The water-sampling device is a box-like dipper narrowed down at the top to a sample receiving slot 1 in. wide by 10 in. long. This opening is large enough to receive any size of solid particles in the water. The size of the container is ample to receive the entire sample without any overflow of water from the container.

3. *Percentage of solids:* Percentage of solids was determined by weighing the entire sample, drying off the water, and weighing the dry solids.

4. *Size consist of solids:* Sizing tests of the residual solids of all the samples were made by wet screening at 270 and 325 mesh and subsequently drying and dry screening the coarse product that remained on the 270 mesh sieve. Subsieve size separations of products of test 2 were made by the Research Division of Heyl and Patterson, Inc., Pittsburgh, Pa., and data are included through the courtesy of that organization.

Micrographs of specimens of some graded fractions in the subsieve size range shown in Fig 4 are of interest in disclosing the particle shapes in the very fine sizes. There are rounded mineral particles among the angular coal particles.

In the statistical treatment of the data, the specific gravity of solids in the water was assumed to be 1.35 in each instance, in order to estimate

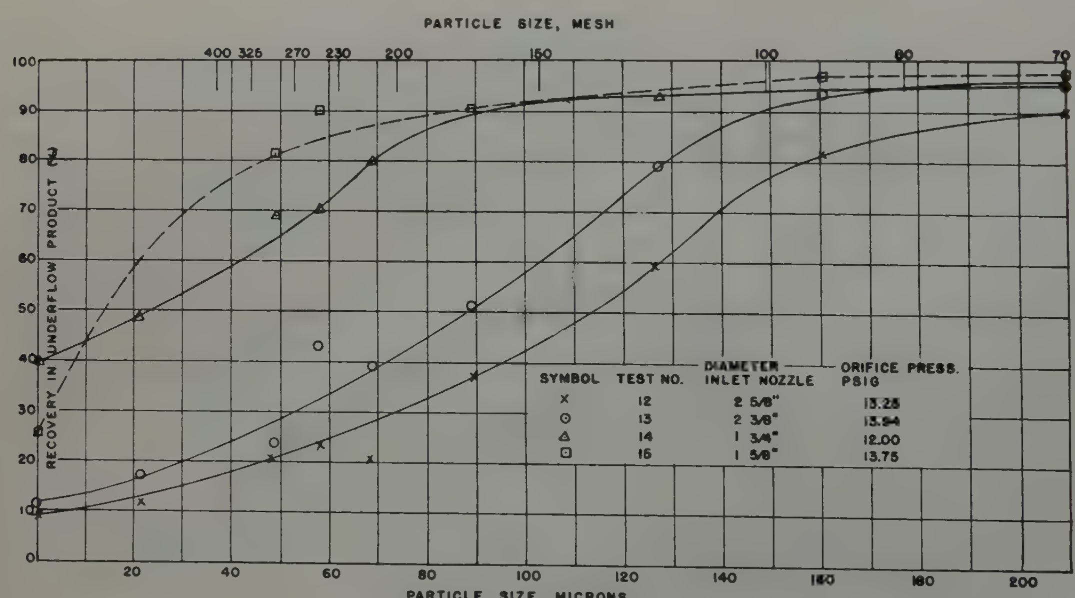

FIG 8—Results of Tests 12 to 15.

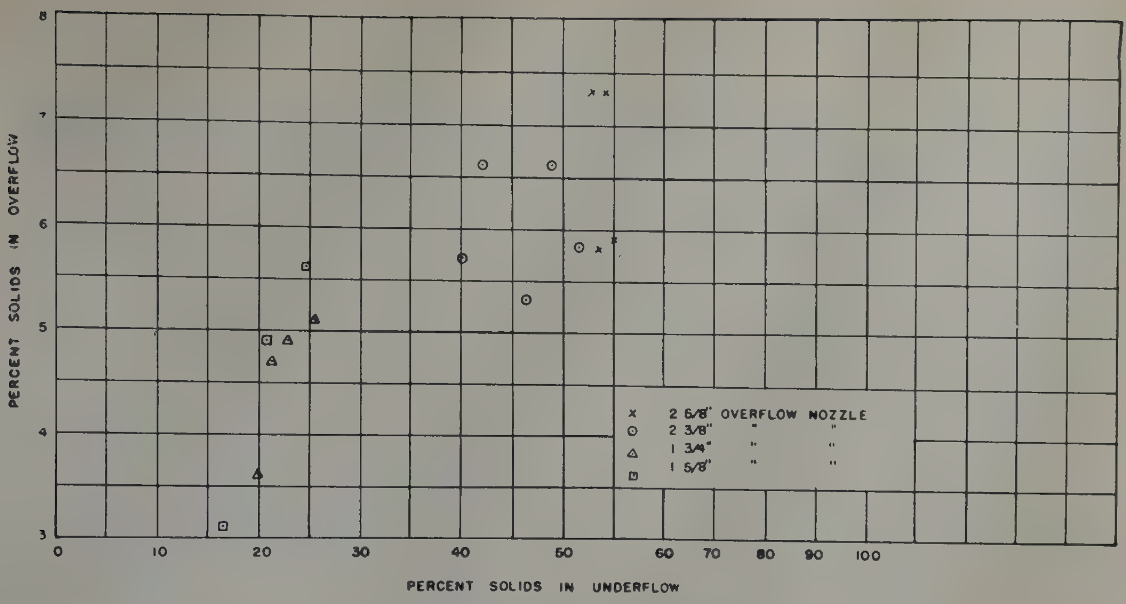

FIG 9—Summary of test data in overflow.

flow, in tons per hour, from the measured data, in gallons. To estimate recoveries in the test runs, the feed to the cyclone unit during the tests was taken to be the sum and composite of the two products, which were directly measured and sampled.

Experimental Data

The performance data obtained in the series of 17 tests of the cyclone thickener are presented in the following recovery charts, in which percentage recovery is plotted against particle size. The complete schedule of test conditions is shown in the insert table of each chart (Fig 5 to 8).

The statistical method of constructing these charts is illustrated by the sample data sheets, Tables 3 and 4, that contain the underlying data of test 1. The percentage recovery is on the weight basis, taking the composite of the overflow plus the underflow to be the feed material.

The throughput of dry solids: These items were determined by computations based on the measured flow in gallons per minute and the percentage of solids, the latter being assumed to have a specific gravity of 1.35. Thus, in test 1, in which the underflow carried 54.1 pct solids, the weight per gallon as measured is 8.337 (the weight of a gallon of water in pounds) divided by $\left(0.459 + \frac{0.541}{1.35}\right)$ or 9.69 lb per gal, and the flow of solids per hour then is $\frac{31 \times 60 \times 9.69 \times 0.541}{2{,}000}$ = 4.875 tons.

The origin point, that is, the percentage recovery shown for 0.0 micron particle size, is a calculated item based on the assumption that the finest particles just go with the water. Hence, the recovery of 0.0 micron size particles in the underflow is the percentage of feed water that goes into the underflow. The figure is obtained by adjusting the flow ratio to correct for solids assumed to have a specific gravity of 1.35 (Table 5).

Table 5 . . . Particles of Subsieve Sizes in Test 2

Size, Microns	Overflow Product, Per Cent		Underflow Product, Per Cent	
	Weight	Ash	Weight	Ash
44 × 30	16.7	4.4	3.7	21.7
30 × 20	17.6	29.7	2.3	21.2
20 × 10	23.4	12.8	1.6	9.1
10 × 0	11.7	21.4	1.4	34.8
Total, −325 mesh	69.4	16.7	9.0	29.5

The data of Fig 9 and Table 6 show the solids in the overflow increased with increased percentages of solids in the underflow. The rate of increase appeared to be greater when the solids in the underflow exceeded 50 pct by weight. With one exception the tests showing more than 50 pct were obtained with a 2⅝ in. overflow nozzle. This nozzle was not shown in the original specifications supplied to the Bureau of Mines by the Dutch State

Table 6 . . . Summary of Test Data

Tests at 2⅝ in. overflow setting to give an O/U[a] area ratio of 3.6.

Test	Pressure	GPM	Recovery	Solids Per Cent in		
				Feed	Overflow	Underflow
1	34.25	252	54.3	13.1	7.3	54.1
16	34.50	264	55.5	11.1	5.9	55.0
5	24.50	209	46.2	12.8	8.1	57.2
8	20.00	198	56.4	11.0	5.8	53.5
12	13.25	163	54.4	13.1	7.3	52.7

Tests at 2⅜ in. overflow setting to give an O/U[a] area ratio of 3.0.

Test	Pressure	GPM	Recovery	Solids Per Cent in		
				Feed	Overflow	Underflow
2	34.75	234	59.9	11.7	5.8	51.6
17	35.00	227	52.6	9.5	5.3	46.3
6	24.50	193	52.7	11.8	6.7	49.8
9	20.00	184	59.8	11.2	5.7	40.0
13	13.94	153	57.4	12.3	6.6	42.0

Tests at 1¾ in. overflow setting to give an O/U[a] area ratio of 1.6.

Test	Pressure	GPM	Recovery	Solids Per Cent in		
				Feed	Overflow	Underflow
3	34.25	188	78.5	9.9	3.6	19.9
7	24.50	162	77.3	13.1	5.1	25.4
10	20.60	148	78.1	12.5	4.9	22.9
14	12.00	124	78.0	11.8	4.7	21.1

Tests at 1⅝ in. overflow setting to give an O/U[a] area ratio of 1.4.

Test	Pressure	GPM	Recovery	Solids Per Cent in		
				Feed	Overflow	Underflow
4	36.0	181	78.8	13.9	5.6	23.7
11	20.0	142	84.6	9.7	3.1	16.4
15	13.75	123	82.9	13.2	4.9	20.7

[a] In this expression (O/U), O represents the area of overflow orifice and U represents the area of underflow orifice.

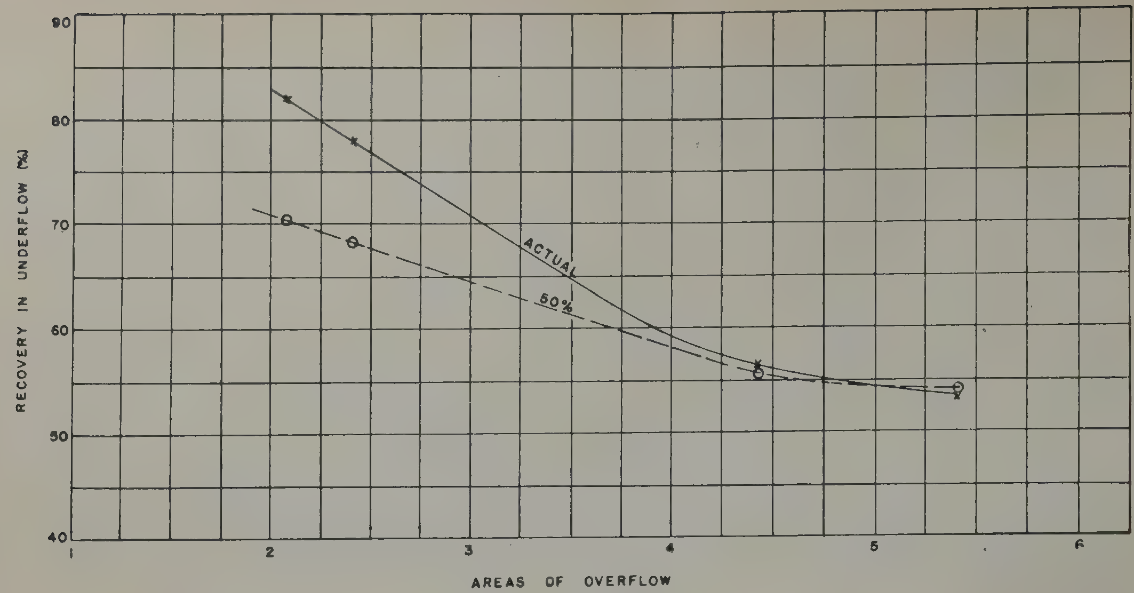

FIG 10—Summary of test data in underflow.

Mines. It was added after the first preliminary tests were made.

It was observed that the vortex action of the underflow discharge decreased as the percentage of solids in the underflow increased and was practically nonexistent when the cone was overloaded.

With any diameter of overflow nozzle used the solids in the overflow increased with increase in the solids in the feed.

Overflows of lower solids content and higher percentage of recovery of solids in the underflow were obtained with overflow nozzles of smaller diameter which produced underflows of lower solid content (Fig 10). However, the higher recovery was more apparent than real. Assuming that the solids in the excess water in the underflow was the same as in the overflow, this amount should be subtracted from the recovery shown. The curve (Fig 10) shows the reduction in percentage of recovery in the underflow if the solids content in the underflow was corrected to 50 pct in all tests. This assumption is not strictly correct. As stated earlier, the data showed there was an increase in solids in the overflow with increase in solids in the underflow. The real recovery curve could, therefore, be expected to fall below the calculated curve shown in the dotted line.

There is also a practical operating objection to an underflow of low solids content—the difficulty of further dewatering the underflow. A practical balance between clarification effected and disposal of underflow appears to be at about 50 pct solids in the underflow. This produced an overflow containing 5 to 6 pct solids with the material treated. Adjustment of relative overflow and underflow nozzle diameters to maintain this condition with an average percentage of solids in the feed should permit operation on a considerable range of feed concentrations without producing an underflow containing objectionable amounts of water or overloading the cone to a point where it would become inoperative.

Conclusions

The only changes in operating conditions with the equipment available were in the diameter of the overflow nozzle and in inlet nozzle pressure. Because of these limitations, it is possible to draw only general and tentative conclusions. No attempt has been made to correlate the results of the tests with those of others experimenting with smaller cones on which more variables could be studied. Essential data covering changes in operating conditions are summarized in Table 6.

The most significant relationship established in these tests is the effect of changes in solids in the feed water going into the cone. At any given orifice setting, a change in the percentage of solids in the feed was reflected in a corresponding change in the solids content of the overflow water. More complete clarification was effected with lower solid contents in the feed.

Some improvement in clarification was obtained with smaller overflow nozzles, with a reduction in capacity.

Throughput was approximately proportional to the square root of the inlet nozzle pressure for any overflow nozzle diameter.

No definite relation between inlet nozzle pressure and clarification was observed for any ratio of overflow to underflow nozzle diameter.

A practical optimum relation between capacity, solid content of underflow and clarification effected was obtained with an overflow nozzle diameter of 2⅜ in. and an underflow nozzle diameter of 1⅜ in., with solid contents of 9.5 to 12.3 pct in the feed. The combination resulted in a reduction in the solid content of the feed of about one-half with an underflow containing about 50 pct solids.

Further tests should be made varying the underflow nozzle diameter to maintain approximately constant solids to determine the effect of varying ratio of overflow-underflow nozzle diameters on clarification.

The series of tests as a whole indicates the cyclone performs fundamentally as a classifier and the major control factor is the flow ratio established by the relation between the overflow and underflow spigot areas. Treating the Shamrock washing water it operated effectively with an orifice ratio of 3, reducing the solid content of the water by approximately half and delivering a thickened underflow of about 50 pct solids.

Titanium Investigations: The Laboratory Development of Mineral-dressing Methods for Arkansas Rutile*

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Introduction

The progress made to date in the mineral dressing of complex Arkansas titanium ores is reported in this paper. Concentrates of rutile, a dioxide of titanium, were produced by treating a submarginal ore from the Magnet Cove Rutile Co., Inc., Magnet, Ark. The rutile recovery at present is 45 to 50 pct of the total TiO_2, with the possibility of increased recovery with additional research.

The work reported herein is part of a general study which will later include results of investigation of other titanium ore bodies in the Magnet, Ark., area.

The fragile character of the rutile minerals, the softness of the gangue, the occurrence of pyrite and leucoxene, and the considerable proportion of extremely small size rutile in the presence of montmorillonite- and illite-type clays in the Magnet Cove open-pit mine are the important factors in the mineral dressing of this submarginal rutile ore.

History

The occurrence of titanium minerals in the Magnet Cove area of Arkansas was recognized as early as 1890, but it was not until about 1912 that any appreciable prospecting for the minerals was undertaken. At that time drilling was begun, a shaft was sunk, and several drifts were driven, but beneficiation to recover the titanium minerals was not attempted.

The only successful mining and milling operation of the Magnet Cove rutile deposit was begun in 1932 by the Titanium Corp. of America and continued until 1942. The ore was washed by means of a rotary scrubber and then deslimed in a drag classifier. The discharge was ground in a rod mill, classified, and tabled to recover a rougher rutile concentrate, which was screen-sized and retabled on cleaner tables. The cleaner concentrates were given a reducing roast, screen-sized, and magnetically separated to produce a finished rutile concentrate. Most of the rutile produced during this period was sold abroad.

In 1942, the Titanium Alloy Corp. of Arkansas (a subsidiary of the Titanium Alloy Manufacturing Co. of New York) purchased the property and rebuilt the mill.

Subsequently, the plant and property were acquired by the Magnet Cove Rutile Co., Inc., of Little Rock, Ark., the present owners. Very little titanium has been produced since 1942.

Although the Titanium Corp. activities yielded enough returns to allow continued operation for approximately 10 years, certain factors must be recognized. These are: (1) production of rutile in that period was during a low in the economic cycle which permitted low-cost operation on a small scale, and (2) a considerable portion of the production was above the water level and no drainage problems existed. Prior to this paper there has been no published report of any mineral-dressing investigation of the lower levels of the ore body. Reference is made in another paper[1] to an earlier investigation of Magnet Cove rutile by the Rolla laboratory, in which it is stated that recoveries of 12 to 17 pct of the rutile were obtained.

The present investigation was undertaken to improve the known milling processes and to provide adequate metallurgical information about the ore treatment.

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Titanium Technology

The titanium mineral of primary interest in the present investigation was the titanium dioxide, rutile. Rutile is one of the industrial minerals whose annual consumption is relatively small but strategically important. The domestic production of rutile was 7100 tons in 1948. The imports of rutile in the same year were approximately 8000 tons.[2]

A major use of rutile is in the production of welding-rod coatings, for which a product containing a minimum of 92 pct titanic oxide is desired. Other constituents, such as lime, silica, and sulphur, must be held to a minimum, but hard-and-fast specifications for these elements cannot be drawn. It is necessary to incorporate the rutile in a welding-rod coating, make a sufficient number of test welds, and then determine the tensile strength of the weld metal in the as-welded and stress-relieved conditions. Such a test gives

[1] References are at the end of the paper.

the behavior of the rutile under operating conditions and is the standard means of determining the usability of an untested rutile. Rutile is also used in manufacturing titanium tetrachloride, in ceramic glazes, and for the production of alloys and the carbide.[3]

Recent advances in titanium technology promise to open a new field for the metal and its alloys in both strategic and structural uses. The combination of low unit weight and high proportional limit gives titanium metal a potential advantage where conditions of high stress are encountered. The Bureau of Mines is continuing primary research in titanium technology, and additional information on this subject may be found in recent publications by Bureau technologists.[4,5]

Description of Ore

The Magnet Cove ore body is near the northeastern part of Hot Spring County, Ark. The small town of Magnet is 2.7 miles east and Hot Springs about 12 miles west.

The ore has been mined by open-cut methods, and that part of the open pit which is exposed at present shows a disseminated formation. The rock is a highly altered mixture of clays, feldspars, carbonates, iron oxides, pyrite, and other minerals. The alteration has proceeded to such an extent that much of the rock is soft enough to be disintegrated by hand pressure.

The Magnet Cove area has been described by Ross and Hendricks[6] as the throat of a volcanic vent. According to these authorities, most of the alteration of the formation has been caused by hot solutions and volcanic vapors. The porous nature of the formation, it is stated, facilitated thorough alteration.

The sample of the Magnet Cove rutile ore for metallurgical investigation was taken from the floor of the open pit by engineers of the Mining and Metallurgical Divisions of the Bureau of Mines. As observation indicated that the upper few inches of material were more claylike than the deeper material, the top 6 in. was discarded. The material from 58 test pits was combined to form one 5000 lb sample, considered to be fairly representative of the ore, which was shipped to Rolla, Mo., for metallurgical work.

The ore, as received at the Rolla laboratory, had an average moisture content of about 10 pct. The material was sampled by coning and quartering and then stored in sacks in covered cans to retain the sample in its original moist condition. This was necessary because much of the clayey rock tended to indurate as it dried. Such dried, hardened material was then more difficult to disintegrate in the subsequent treatment.

The sample was composed principally of altered albite and clay minerals of the montmorillonite and illite groups, with appreciable partly altered orthoclase and ankerite. Minerals present in small to very small amounts included pyrite, rutile, calcite, phlogopite and other micas, apatite, goethite and limonitic iron oxides, leucoxene, sanidine, gypsum, chert, sericite and sphalerite.

The titanium minerals were rutile and leucoxene. Most of the rutile was a black variety, which occurred in several forms. One of them, shown in Fig 1, consisted of veinlets of striated crystalline masses and closely packed needles. Much of the rutile was found as small crystalline grains and in scattered bunches containing from a few needlelike crystals up to thickly clustered, acicular groups. The latter occurrence is well-illustrated by Fig 2. Fig 3 is a photograph of grains that were acid-leached to remove the ankerite to give a better picture of the felted mass of tiny acicular rutile crystals. In addition to the occurrences noted, rutile was also found disseminated throughout many rock types and as linings in cavities.

The mineral leucoxene was rather difficult to identify because of its indeterminate optical properties. It was an important constituent of the clayey fraction of the ore, particularly the hydrous mica. In some instances tiny bunches of rutile needles were partly altered to leucoxene. A micrograph of a thin section, showing leucoxene and rutile in ankerite and albite, is shown in Fig 4.

With regard to liberation, much of the rutile was free at minus 20 mesh, but a considerable quantity was still locked in sizes finer than 200 mesh. Some of the tiny needles of rutile previously noted were less than 1 micron in width.

The occurrence of some of the other minerals was of interest, particularly as it affected the ore-dressing treatment. The pyrite ranged from blocky, euhedral crystals more than 1 in. thick

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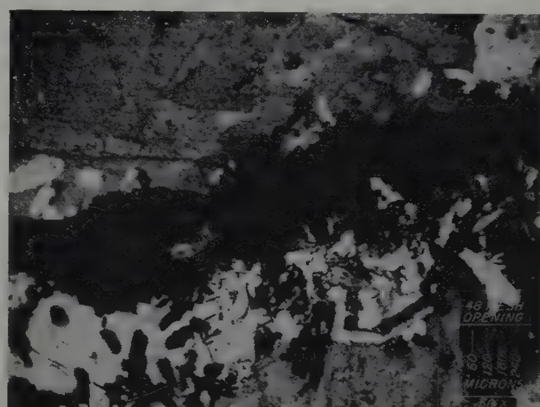

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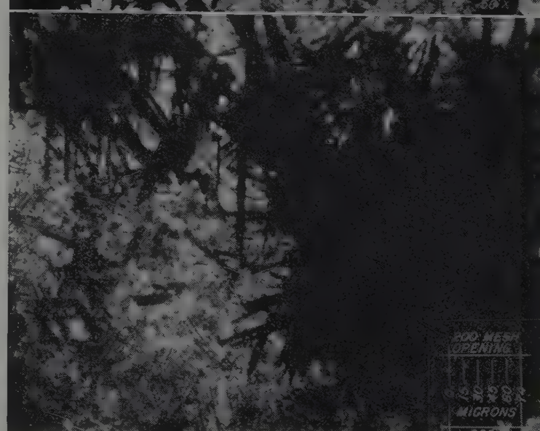

FIG 1—Veinlet of rutile in ankerite (gray and white), with albite (white). Nicols crossed.
FIG 2—Clusters of rutile in albite and orthoclase. Apatite and ankerite are present. Nicols uncrossed.

to tiny grains disseminated throughout the mass. In general, however, the pyrite was more coarsely crystalline than the rutile. Apatite was present in virtually all rock sections as small, scattered, lath-shaped crystals. A concentration of these crystals was observed in the section shown in Fig 5.

In addition to the usual optical microscopic observations, the presence of some minerals was confirmed by differential thermal analyses. A group of clay fractions was examined by this means, and the thermal curve of one such fraction is the upper one of Fig 6. This product gave pronounced thermal reactions characteristic of the montmorillonite group and weak reactions for illite. A differential thermal analysis of ankerite (the complex calcium-magnesium-iron carbonate) is the lower curve of Fig 6. Except for small shifts in the temperature of the reactions, this curve showed a striking resemblance to the thermal analyses of the minerals siderite and dolomite.

The sample had the following chemical analyses: 3.4 pct titanium dioxide; 7.4, iron; 33.6, silica; 13.4, alumina; 13.7, calcium carbonate; 4.1, magnesia; 3.1, sulphur; 0.64, phosphorus; and 0.06, vanadium pentoxide. In addition, water-soluble calcium and sulphate were present to the extent of 0.21 and 0.52 pct, respectively.

It must be noted at this point that not all the titanium in the ore, as determined by a chemical analysis for that element, was available as rutile. The leucoxene represented an important (but indeterminate) fraction of the total titanium which was not recoverable by other than chemical means. In addition, the numerous rutile grains whose largest dimensions were measurable in microns or fractions thereof were not regarded as recoverable by the usual ore-dressing methods. In consideration of these factors, it was estimated that a maximum of perhaps 50 to 60 pct of the titanium dioxide would be recoverable by known ore-dressing methods.

Beneficiation Studies

DISINTEGRATION AND DESLIMING

The first step in milling the Magnet Cove ore was disintegration of the clay and semiplastic constituents and subsequent removal from the granular material. For that purpose, a continuous test was made in equipment of pilot-plant size to remove the clay and recover granular products for subsequent treatment. About 2500 lb of the ore, crushed to minus 1½ in., was treated at the rate of 180 lb an hour according to the following flowsheet: the ore was fed to a 3 ft long cylindrical mill with an effective diameter of 15 in. and an 8 in. discharge operated at 44 rpm. No grinding medium was used, so the action was one of attrition of ore particles upon each other. Some of the larger, heavier rocks were retained in the mill longer than the body of the pulp, and these aided in disintegration of the softer fragments. The pulp density in the mill was 50 to 55 pct solids. A cylindrical screen with ⅜ in. square openings was fitted in the discharge end of the mill, and particles coarser than that size were washed and removed as the pulp discharged from the mill.

The screen undersize was laundered to a 12 in. spiral classifier, where a coarse sand was separated. The spiral overflow was pumped to a 3 ft bowl classifier, where a fine sand was recovered and the slime product was sent to waste. The results of the desliming are noted in Table 1.

Table 1 . . . Desliming

Product	Weight, Per Cent	Analysis, Per Cent		Per Cent of Total	
		TiO_2	Fe	TiO_2	Fe
Plus ⅜-in. sand	17.5	3.6	8.4	19.3	17.7
Spiral-classifier sand	38.5	4.9	11.5	57.8	53.2
Bowl-classifier sand	8.4	2.1	5.5	5.4	5.6
Primary slime	35.6	1.6	5.5	17.5	23.5
Composite	100.0	3.3	8.3	100.0	100.0

The material rejected by the bowl classifier, the primary slime, was lower-grade than the feed and contained 17.5 pct of the titania in 35.6 pct of the weight. It was very fine and contained only 3.3 pct of plus 325 mesh material.

PRIMARY GRAVITY CONCENTRATION

The plus ⅜ in. material was crushed in a rolls crusher to pass that size and combined with the spiral-classifier sand. The combined product was screened on 4, 10, and 20 mesh sieves to separate sized products suitable for jigging. All sizes coarser than 20 mesh

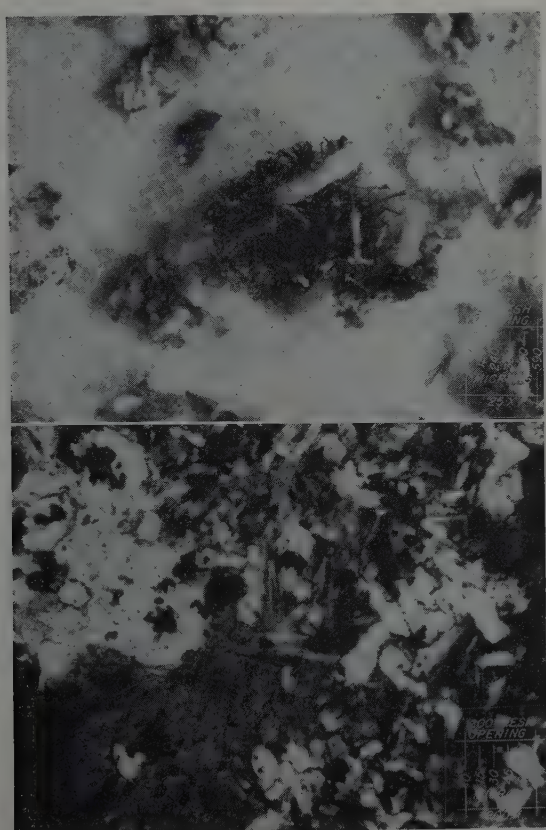

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FIG 3—Minus 10 plus 28-mesh acid-treated grains for removal of ankerite to illustrate an occurrence of rutile in finely acicular aggregates. Two feldspar crystals are shown in the mass.

FIG 4—Ankerite (dark gray, in foreground), albite (white and gray areas), and leucoxene and rutile (black areas). Nicols crossed.

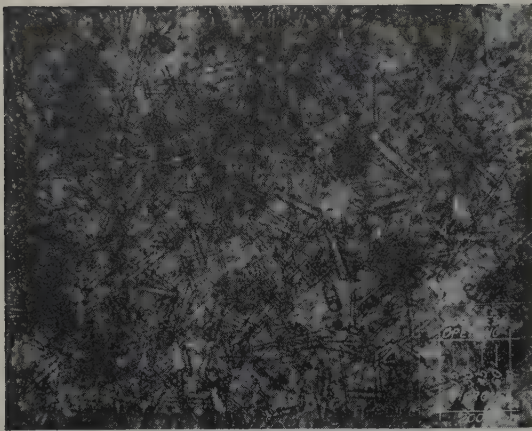

FIG 5—Matted mass of apatite crystals in partially altered albite and orthoclase. Ankerite and rutile were present in the thin section. Nicols uncrossed.

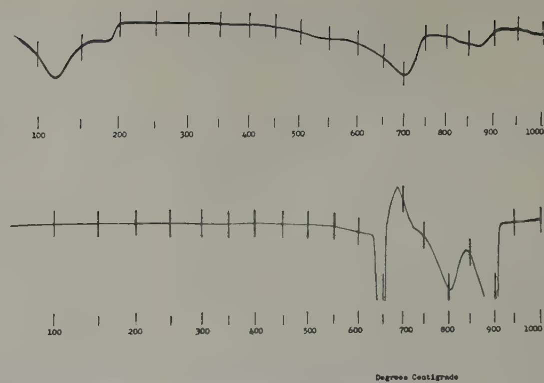

FIG 6—Differential thermal analysis curves of clay and ankerite from Magnet Cove rutile pit, Magnet, Ark.

Top: water suspended fraction of a white claylike material containing much residual albite. *Bottom:* ankerite (massive type).

were jigged separately in a laboratory Harz-type jig to produce concentrates, middlings, and tailings. Each middling was crushed and added to the feed at the next finer size.

The jig concentrates were mixtures of rutile and pyrite, with about 10 pct of ankerite and feldspar gangue, some of it locked with rutile. The pyrite, however, was for the most part free, so the possibility of removing some of it by gravity was explored.

Unfortunately, the amount of mixed rutile-pyrite concentrate was insufficient to try the separation over a full range of sizes so a compromise was necessary. The rutile-pyrite mix was crushed to minus 4 mesh and sized on 8 and 20 mesh sieves. Each size was jigged separately on the same Harz-type jig to remove as much clean pyrite as possible. All the pyrite concentrate was drawn through the screen into the hutch, as this gave a cleaner pyrite than a cup-drawn product. On the 4 to 8 mesh size, 44.5 pct of the iron was removed in a pyrite product containing 0.6 pct titania and 43.3 pct iron. Only 1.4 pct of the titania in this fraction was lost in the iron product. The 8 to 20 mesh size did not separate as well as the other, but 23.7 pct of the iron in this fraction was removed as a pyrite jig concentrate containing 1.2 pct titania and 43.8 pct iron.

In both sizes the blocky, euhedral crystals of pyrite reported in the concentrate without much difficulty. Much of the pyrite, however, was tabular and as a result had an apparent specific gravity in the jig very nearly that of the rutile. Such pyrite was, of course, not separable from rutile by gravimetric means. The pyrite was so hard, however, that it was desirable to remove as much of it as possible at coarse sizes.

The middling from the finest jig size, the primary minus 20 mesh, and the bowl sand were combined for table feed. The composite was hydraulically classified into four sand fractions and a slime, and each sand was tabled separately to produce a rutile-pyrite concentrate, middling, and tailing. Most of the pyrite, of course, came into the concentrate zone of the table readily. Some of the elongated rutile crystals, however, were not as responsive to the action of the table and reduced the effectiveness of the separation. The net result was a fairly large middling containing an appreciable titanium content in addition to ankerite and apatite.

A composite of the results of jigging and tabling is given in Table 2. The primary slime is also included to show a complete picture of the beneficiation up to this point.

Table 2 . . . Jigging and Tabling

Product	Weight, Per Cent	Analysis, Per Cent		Per Cent of Total	
		TiO_2	Fe	TiO_2	Fe
Jig concentrate..	3 89	22.9	31.9	26.1	15.6
Table concentrate..........	2.82	24.3	27.5	20.1	9.7
Pyrite concentrate.........	1.24	0.8	43.5	0.3	6.8
Middling........	0.95	14.8	10.2	4.1	1.2
Jig tailing.......	27.84	1.8	5.9	15.1	20.8
Table tailing....	25 11	2.1	6.3	15.4	19 8
Table slime.....	2.58	2.9	4.8	2.2	1.6
Primary slime...	35.57	1.6	5.5	16.7	24.5
Composite....	100.00	3.4	8.0	100.0	100.0

A total of 46.1 pct of the titania was collected in the form of rutile-pyrite gravity concentrates plus an additional 4.1 pct in the middling. The tailings accounted for 30.5 pct, which is a substantial part of the total.

Much of the titanium in the tailing was, as previously noted, in the form of extremely fine needles of rutile, which would require reduction to micron sizes for complete liberation. Part of the rutile, however, was sufficiently coarse so that only a moderate degree of comminution would produce adequate liberation. To improve the recovery, it was decided to retreat the tailings from jigging and tabling. The treatment was in the nature of a scavenging operation to produce a low-grade concentrate which could be returned to the table circuit for additional treatment. The Humphreys spiral concentrator was utilized for this purpose because of its capacity to handle a large tonnage of material at low cost.

A composite of the jig and table tailing was ground to minus 48 mesh and fed to the spiral concentrator in a pulp of about 15 pct solids. The machine was adjusted so that concentrates were withdrawn from six ports and middlings from two. The middlings were recirculated, and the rest of the pulp constituted the tailing. Both concentrates and tailings were sampled for timed intervals, and then an equal weight of new feed was added to the system. After equilibrium was reestablished, another set of samples was taken. The process was repeated often enough to obtain a representative group of samples for further work. The results of a spiral-concentration test are detailed in Table 3.

The recovery of titania was 27.8 pct of that in the feed, or 8.7 pct of the total in the ore. This low recovery was due primarily to intimate interlocking of gangue and rutile. However, the amount recovered represents an important contribution to the total

Table 3 . . . Spiral Concentration

Product	Weight, Per Cent	Analysis, Per Cent		Per Cent of Total	
		TiO_2	Fe	TiO_2	Fe
Concentrate	7.2	8.3	9.5	27.8	11.8
Tailing	92.8	1.6	5.6	72.2	88.2
Composite	100.0	2.2	5.9	100.0	100.0

Table 4 . . . Final Concentration

Product	Weight, Per Cent	Analysis, Per Cent						Per Cent of Total	
		TiO_2	Fe	SiO_2	CaO	S	P	TiO_2	Fe
Coarse concentrate	1.47	93.6	1.3	0.61	0.24	0.09	0.16	38.66	0.24
Fine concentrate	0.32	85.5	1.9	0.68	1.8	0.10	0.86	7.69	0.08
Middling	1.72	17.1	8.5					8.26	1.86
Magnetic	0.08	22.2	35.1					0.50	0.36
Flotation pyrite	1.71	1.2	42.8					0.58	9.31
Agglomeration pyrite	3.10	0.6	42.8					0.52	16.88
Jig pyrite	1.24	0.8	43.5					0.28	6.86
Secondary table tailings	2.96	2.8	7.5					2.33	2.82
Spiral tailings	49.12	1.6	5.6					22.08	34.99
Secondary slime	2.71	4.1	5.0					3.12	1.72
Primary slime	35.57	1.6	5.5					15.98	24.88
Composite	100.00	3.6	7.9					100.00	100.00

Table 5 . . . Concise Tabulation of Results

Product	Weight, Per Cent	Analysis, Per Cent						Per Cent of Total	
		TiO_2	Fe	SiO_2	CaO	S	P	TiO_2	Fe
Concentrate	1.79	92.2	1.4	0.63	0.52	0.09	0.29	46.35	0.32
Middling	1.72	17.1	8.5					8.26	1.86
Magnetic	0.08	22.2	35.1					0.50	0.36
Pyrite	6.05	0.82	42.9					1.38	33.05
Tailing	52.08	1.7	5.7					24.41	37.81
Slime	38.28	1.8	5.4					19.10	26.60
Composite	100.00	3.6	7.9					100.00	100.00

recoverable rutile.

The grade of the concentrate was quite low, as no attempt was made to produce a high-grade material. It is expected that, if spiral concentration is utilized in a plant, the concentrate would be returned to the tables for additional beneficiation.

FINAL CONCENTRATION

At this point in the process, rutile had been recovered by jigging, tabling, and spiral concentration in the form of low- to medium-grade concentrates containing pyrite, ankerite, and some apatite in addition to rutile. The next step in the process was removal of the remaining pyrite. The jigging, previously described, had been effective in removing part of the pyrite but could not effect the nearly complete separation required. In addition, the process was not applicable to the fine sizes.

A combination of agglomerate tabling and flotation was a very effective means of removing the pyrite. The process was simple and gave almost complete removal of the sulphides.

A composite was prepared for this work, which consisted of the jig, table, and spiral concentrates, and the table middling. The composite had a calculated analysis of 17.7 pct titania, 21.6 pct iron, and represented 59.8 pct of the total titanium and 31.0 pct of the iron. Owing to inclusion of such products as the spiral concentrate and table middling, the composite was of lower grade than would be expected in an operating plant where provision could be made for circulating such products. Under the circumstances, however, the composite noted above was the most satisfactory.

The sample was crushed to minus 14 mesh and screened on a 48 mesh sieve as a preliminary step to the removal of pyrite by agglomerate tabling. The 14 to 48 mesh sand was mixed for a few minutes in very thick pulp with 2.0 lb of sodium sulphide to clean the surface of the pyrite. The sodium sulphide then was washed out, and the sand was conditioned with 1.0 lb of sulphuric acid, 0.10 lb of pentasol amyl xanthate, and 10.0 lb of No. 3 fuel oil to produce a water-repellent oil coating on the pyrite. The conditioned mix was then fed to a laboratory shaking table, where the pyrite was removed by filming and agglomeration. The pyrite filmed readily, and no difficulty was experienced in recovering a clean pyrite essentially free of rutile. The unoiled fraction, consisting of the rutile and some gangue material, was collected at the concentrate end of the table. It was then screened (wet) on 20 and 35 mesh sieves, and each size was retabled to produce a coarse rutile concentrate, middling, and tailing. The middling was ground to minus 48 mesh and set aside for further treatment noted below.

The pyrite in the minus 48 mesh sand was removed by flotation with a suite of reagents similar to those used in the agglomerate tabling. The minus 48 mesh sand was pulped to about 40 pct solids, conditioned with 0.50 lb of sodium sulphide, and floated with the following reagents: 0.30 lb of pentasol amyl xanthate, 0.06 lb of Aerofloat 31, and 0.06 lb of pine oil. The flotation pyrite was sufficiently free of rutile so that no cleaning was necessary. The pyrite formed a heavy froth, free of mechanical inclusions, and floated with reasonable rapidity.

In addition to pyrite, the fine fraction contained some sphalerite. The amount of zinc was not of economic proportions but was high enough to influence the sulphur content of the product if it were not removed. The sphalerite was reluctant to float without activation, so additional treatment was necessary. This consisted of conditioning with 2.0 lb of copper sulphate per ton of feed and floating with 0.2 lb of pentasol amyl xanthate and 0.06 lb of cresylic acid. The small amount of zinc concentrate was added to the pyrite product without cleaning.

The nonfloat was combined with the crushed middling from tabling the coarse fraction, hydraulically classified, and tabled to produce a fine rutile concentrate, middling, and tailing. Both the coarse and fine rutile concentrates were dried and passed over a high-intensity magnetic separator to effect a final improvement in the grade. The results, showing all products, are compiled in Table 4, and a more concise tabulation combining similar products is shown in Table 5.

The coarse concentrate was higher-grade than the fine concentrate; but, as shown in Table 5, the composite of the two exceeded the minimum required TiO_2 content of 92 pct. The recovery of titanium in the concentrates shown was 46.35 pct. By the usual ore-dressing standards, this would not be considered good recovery. However, if one considers the quantity of titanium present as leucoxene and as needles of rutile 1 micron or less in width, it is seen that a major portion of the available rutile was recovered.

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Table 6 . . . Middling Flotation

Product	Weight, Per Cent	Analysis, Per Cent					Per Cent of Total		
		TiO_2	Fe	CaO	S	P	TiO_2	Fe	CaO
Rutile concentrate	17.0	70.1	2.3	0.6	0.26	0.16	69.8	4.4	0.5
Carbonate concentrate	71.2	3.0	8.9	25.6			12.3	71.9	95.3
Magnetic	2.4	13.5	28.7	7.0			1.9	7.8	0.9
Tailing	5.7	35.6	18.6	1.4			11.9	12.0	0.4
Slime	3.7	18.8	9.4	15.1			4.1	3.9	2.9
Composite	100.0	17.1	8.8	19.1			100.0	100.0	100.0

the Rolla Branch, Metallurgical Division, is investigating utilization of these rutile concentrates as a constituent of welding-rod coatings. Preliminary results of that study have been promising; complete data will be reported in a later publication.

MIDDLING FLOTATION

The final middling contained an additional 8.26 pct of the titania. The material was, in some respects, a more complex mixture than the feed, and its treatment by any process presented a problem. Its mineralogical composition was similar to the feed; and it contained, in addition to the rutile, ankerite, and other carbonates, feldspars, quartz, iron oxides, and small amounts of apatite and pyrite. However, more of the rutile was present in locked grains than in the heads, and the product was much finer. Therefore, the possibility of treating such material by flotation was considered.

As regards flotation behavior, the mineral rutile, under favorable conditions, is responsive to either the anionic reagents (fatty acids and soaps), or the cationic reagents (long-chain aliphatic amines). Investigations with both types of reagents have been made, and others are in progress, but recovery of the rutile in a satisfactory concentrate has not been achieved. However, the work has shown promise that flotation of selected products containing rutile may have a definite place in the flowsheet for this ore.

The results of one test on the middling is given below. The sample was ground for a short time in a pebble mill to freshen the mineral surfaces, and the pulp was dispersed with 2.0 lb of sodium silicate per ton of feed and deslimed. The sand was conditioned with 3.0 lb of silicate to depress the rutile while floating the gangue carbonates (largely ankerite) with sodium oleate. The ankerite was floated in stages to reduce the amount of rutile mechanically entrapped in the carbonate froth. In all, 4.2 lb of sodium silicate and 2.8 lb of sodium oleate were used to condition and float the carbonates.

At the conclusion of the carbonate float, the pulp was thickened to remove the anionic reagents, and the rutile was floated with 0.25 lb of the long-chain amine ARMAC TD. The froth was largely rutile contaminated with quartz, silicates, small amounts of carbonates, and iron oxides. The concentrate was dried and passed over a high-intensity magnetic separator to effect an additional improvement in grade. The results of the test are presented in Table 6.

The rutile concentrate recovered 69.8 pct of the titania in the feed and had an analysis of 70.1 pct titanium dioxide. The separation of carbonates in this scheme was almost complete and the removal of iron was good. Separation from silicates, particularly feldspar, was incomplete and was primarily responsible for the lowered grade of the concentrate. Research work on the flotation of rutile from complex mixtures, as those described in this paper, is in progress, and any improved techniques will be used to increase the recovery of rutile from the Magnet Cove ore.

Summary and Conclusions

One phase of an extensive investigation of the ore-dressing characteristics of titanium-bearing ores of the Magnet Cove area of Arkansas has been completed, and the results are presented in this paper. A sample of the complex ore from the property of Magnet Cove Rutile Co., Inc., Magnet, Ark., containing rutile, leucoxene, pyrite, ankerite, quartz, apatite, feldspars, iron oxides, clay, and other minerals, was the subject of the investigation.

The treatment process included disintegration of the ore by tumbling in a revolving cylinder and subsequent removal of clay by classification. Gangue minerals and some pyrite were removed by gravity concentration. The rest of the pyrite was removed from the coarse sizes by agglomerate tabling and from the fine sizes by flotation. A finished rutile concentrate was then recovered by sizing, classification, tabling, and magnetic separation. This treatment gave a recovery of 46.35 pct of the titanium in a product containing 92.2 pct titania, 1.4 pct iron, 0.63 pct silica, 0.52 pct lime, 0.09 pct sulphur, and 0.29 pct phosphorus.

The investigation has shown that the production of commercial-grade rutile concentrates from the Magnet Cove property is technically feasible. The best possibility for improving the recovery obtained to date is in extending the application of flotation to the benefaction of gravity middlings and similar products. Those flotation tests which have been made on such materials (one of which is given in this report) show a fair degree of concentration. The flotation study, while not productive of very high-grade concentrates, has been promising enough to warrant further work. A method of utilizing noncommercial grade rutile concentrate for the production of titanium matte is being investigated and the preliminary results are reported in another paper.[7]

Acknowledgments

Acknowledgment is made to the late R. G. O'Meara, metallurgist of the Rolla Branch, for helpful advice during the progress of the investigation.

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Guide for Buying Domestic Muscovite Mica

By BLANDFORD C. BURGESS,* Member AIME

Introduction

Mica is an orchid among minerals. It is formed in pegmatites, one of the most bizarre of igneous formations, and is exceeded by few other minerals in the perfection it may attain as to size, color, and cleavage.

First used for window glazing, as an ornament and article of trade, it has become one of the most essential insulators of electricity, heat, and shock. Most internal combustion engines use mica condensers or capacitors and such machines of war as planes, tanks, and submarines are immobile without it.

The original manuscript of this guide was written in April 1943 at the peak of war emergency. It was not used because shortly thereafter a decision was made to buy on a one price basis eliminating differentials for both quality and size.

This is now presented with the thought that from the standpoint of preparedness we can only consider another war imminent and every effort should be made to set up in usable form the experience gained during the Second World War.

Colonial Mica Corp. was set up by Metals Reserve Co. as a wholly owned government agency to encourage production of domestic mica and to purchase it. It soon became the exclusive buying agency for such mica. The writer as Southern Manager called on all the private agencies buying mica to assist in formulating buying policies, methods, and prices.

A meeting was held for that purpose in Asheville, N. C., July 20, 1942. In the general discussion the first two hours of the meeting it was decided that the following factors should be considered in pricing strategic mica: clearness, flatness, trimming, color, classification (size and quality), free splitting, hardness, and air inclusions.

Most of the mica miners have been buying strategic mica, according to the mine, as No. 1, No. 2, or Nos. 1 and 2 combined.

In addition to these grades of strategic mica they have been buying three lower grades of mica, electric (vegetable-stained), black-spotted and black-stained.

A number of mines were named one by one and the buyers present gave their opinions as to the basis they would use in buying the sheet mica from each.

It was suggested that in addition to the prices already published, prices be posted up to 10 pct higher for special quality and preparation and prices be posted 25 pct lower than the published prices for the lowest acceptable grades of strategic mica, giving the buyers authority to make intermediate prices according to the quality and preparation.

It was thought desirable at the same time to announce prices for nonstrategic mica where such quality is being obtained in mines producing strategic mica and where it would be necessary to take the output in order to get the strategic mica.

It was also suggested that the price of electric and black-spotted mica be 50 pct below the price of the lowest grade of strategic mica and the price of black-stained mica be 60 pct below the price of lowest grade strategic mica.

The few experienced buyers available were employed but they were inadequate for the job and others had to be trained. All needed a guide or primer of some kind to attain systematic procedure and consistent prices in their respective buying areas. The Mica Buyers Guide was formulated for that purpose based on experience gained in the first six months of buying.

The Preface to the original guide was as follows:

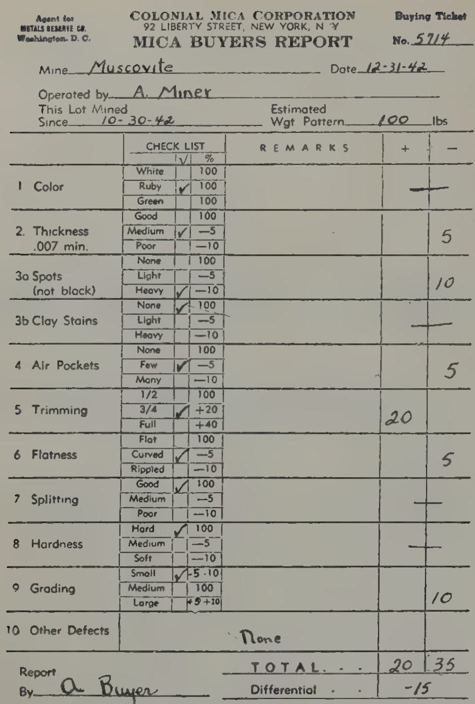
> The purpose of this guide is to serve as a text in training beginners and as a handbook for experienced buyers of the instructions which have been issued up to this time. At the meeting of prospective buyers last summer, it developed that the first essential to attain uniformity must be some kind of check sheet. Our "Mica Buyers Report" was developed for this purpose and forms the backbone of this guide. Price is determined according to size, quality and preparation. Size can be measured, but no system of measurement or machine has been devised which can be substituted for the trained eye of the buyer to determine quality and preparation.

Although the "One Price" system was instituted and the "Guide" not used, the buyers were required to continue making out a "Mica Buyers Report" (Fig 1) with each lot of mica purchased. The form proved its value and there is no doubt that in another such emergency some similar form should be

San Francisco Meeting, February 1949.

TP 2669 H. Discussion of this paper (2 copies) may be sent to *Transactions* AIME before Jan. 31, 1950. Manuscript received Nov. 3, 1948.

* Consulting Mining Engineer, Monticello, Georgia.



Agent for METALS RESERVE CO. Washington, D. C.

COLONIAL MICA CORPORATION
92 LIBERTY STREET, NEW YORK, N Y
MICA BUYERS REPORT

Buying Ticket No. 5714

Mine Muscovite Date 12-31-42

Operated by A. Miner

This Lot Mined Since 10-30-42 Estimated Wgt Pattern 100 lbs

	CHECK LIST	✓	%	REMARKS	+	−
1 Color	White		100			
	Ruby	✓	100			
	Green		100			
2. Thickness .007 min.	Good		100			
	Medium	✓	−5			5
	Poor		−10			
3a Spots (not black)	None		100			
	Light		−5			
	Heavy	✓	−10			10
3b Clay Stains	None	✓	100			
	Light		−5			
	Heavy		−10			
4 Air Pockets	None		100			
	Few	✓	−5			5
	Many		−10			
5 Trimming	1/2		100			
	3/4	✓	+20		20	
	Full		+40			
6 Flatness	Flat		100			
	Curved	✓	−5			5
	Rippled		−10			
7 Splitting	Good	✓	100			
	Medium		−5			
	Poor		−10			
8 Hardness	Hard	✓	100			
	Medium		−5			
	Soft		−10			
9 Grading	Small	✓	−5 −10			10
	Medium		100			
	Large		+5 +10			
10 Other Defects				None		
				TOTAL . . .	20	35
				Differential . .	−15	

Report By A Buyer

FIG 1—Mica buyers report.

used to attain a uniform standard system of buying domestic muscovite mica.

This paper is presented as the first effort to set up a basis for such a system.

Mica Buyers Guide, Definitions

"A" mica, mica with converging reeves.

Air pockets, air inclusions between layers of mica (see Mica Buyers Report 4, p. 456).

Beryl, a pegmatite mineral, ore of beryllium. Emerald.

Biotite, black mica, contains iron and magnesia.

Block, as used by importers and foreign mica workers—called sheet mica by domestic miners and buyers. As used by domestic miners, the blocks or books of mica sorted from the run-of-mine mica.

Chlorite, a green mica-like mineral usually in small flakes.

Class, to sort sheet mica as to size and quality. So termed by Domestic Miners and Buyers.

Cobbed mica, used in New England to describe mica books freed of obvious scrap ready for rifting.

Columbite, a pegmatite mineral, ore of columbium.

Condenser, a device to provide electrical energy capacity.

Condenser films, sheet mica from good quality full trim split to thickness suitable for condensers, usually 1 to 4 mils.

Cross-grained, mica which, due to structure, or imperfection will not split freely into sheets, or film of uniform thickness. Referred to by domestic miners as "Gummy."

Diaphragm mica, suitable for use in phonograph diaphragm (obsolete). Now used in some types of respirators, must be perfectly flat, but other features not important.

Dielectric strength, resistance to puncture by electric current.

Electric mica, black-stained and heavy black-spotted mica (domestic miner's term).

Feldspar, spar, a silicate of alkalies and alumina. One of the three principal pegmatite minerals.

Films, sheet mica split to thickness and of quality suitable for condensers.

Grade, to sort sheet mica as to size (see also Class).

Hair lines, fine cracks (see under Defects p. 457).

Hardness of mica, 2 to 2½, according to Moh's scale.

Inclusions, other mineral or substance between layers of mica.

Isinglass, old name for mica.

Knife trimmed, mica trimmed by hand with a beveled edge (K.T.). Method desired for strategic mica, because beveled edge aids in splitting.

Lepidolite, lithia mica.

Mica, a group name for a number of aluminum silicate minerals, one of the three principal pegmatite minerals. It is characterized by high reflectance and perfectly developed basal cleavage; tough, elastic, transparent, and of high electrical resistance.

Muscovite, potash mica, the principal commercial mica.

Pattern mica, sheet mica as termed by domestic producers or block as used by importers. To cut a standard size pattern.

Pegmatite, igneous rock consisting chiefly of feldspar, quartz, and mica with great variation in the size of the crystals and some grouping of the minerals. The host rock for mica.

Phlogopite, amber or magnesia mica. Quality judged according to test for heat resistance as "high heat" or "low heat."

Pitchblende, black uranium mineral occasionally found with mica.

Power factor, power loss under electric load.

Punch mica, (see p. 455.) Thumb trimmed sheet mica with clear, solid area less than 1½ by 2 in.

Quartz, silica (SiO_2) usually called flint by mica miners. One of the three principal pegmatite minerals.

Qualify, sort mica as to quality.

Reeves, domestic term for a series of tiny ripples or waves, which sometimes cause a cross-grained structure.

Ribbon mica, mica with straight, parallel, edges, sheared by earth movements, etc., of varying widths, and very desirable for certain purposes.

Rifting, splitting cobbed mica or books or blocks of mica and trimming the sheet or pattern. "Should be defined only as splitting to proper thickness for trimming." Sometimes both operations are performed by one operator, but strictly speaking rifting and trimming are two operations performed by different workers.

Rifting shop, shop where rifting and trimming is done.

Ruled mica, straight ridges or cracks in mica, often parallel (see Ribbon mica). Ruled mica generally refers only to mica completely sheared into strips of uniform width.

Ruling, straight ridges or cracks (see Ribbon and Ruled mica).

Run-of-mine mica, all the mica obtained, generally including the scrap, but this should be stated.

Scrap, mica not suitable for punch or pattern or trimmings from punch and pattern. Also called waste and refuse. Generally classified as "mine scrap" and "shop scrap," shop scrap bringing premium price.

Sericite, mica occurring in a fine flaky form (secondary mica).

Shear trimmed, trimmed with shears gives a square edge compared to the beveled edge of knife trimmed. (Not rejected, but this method should be discouraged in preference to knife trim, to conserve mica and facilitate splitting.)

Skimmings, thin films of mica split from sheets or blocks to remove imperfections (untrimmed thins).

Specific gravity of mica, 2.76 to 3.00 (times equal volume of water).

Splittings, sheet mica split to thickness

suitable for condensers (see also Condenser splittings). Used for manufacture of mica plate, or built up mica. (Not usually suitable for condenser film.)

Tantalite, a pegmatite mineral, ore of tantalum.

Tests, see dielectric, power factor, etc.

Thins, sheet mica less than 0.007 in. thick. Distinguished from skimmings by being ½, ¾. or full trimmed and priced by preparation and quality.

Thumb trimmed. rifted but trimmed only to extent rough edges can be broken off with fingers. The usual preparation of punch mica.

Trimming, removing rough edges, inclusions, and other imperfections from rifted sheet mica, with knife or shears.

Vermiculite, a form of mica which exfoliates on exposure to heat. Used exfoliated for heat and sound insulation and lightweight aggregate.

Washer mica, a quality of mica between punch and scrap. Punch mica of No. 3 quality (cross-grained or heavy stained) with solid area of 1½ in. diam upward.

Wedge mica, books of mica thicker on one edge than on the opposite edge. These frequently also have "A" structure.

Zinnwaldite, a mica containing potash, lithium, and iron. Poor splitting. (Quite rare, some at Amelia, Va.)

Prices

The base prices of Table 1 were made effective Nov. 1, 1942 and were guaranteed to Dec. 31, 1943.

Table 1 . . . Colonial Mica Corporation Buying Price Schedule for Domestic Strategic Mica
(Revised Jan. 6, 1943)

Table A		Table B	
Punch	30¢ Per Lb	Sheet—Grades to Cut Mininum of:	
Sheet—Grades to Cut Minimum of:			
	Per Lb		Per Lb
1½ × 2[a]	$2.40	1 × 1[a]	$1.00
2 × 2	3.52	1¼ × 1¼	1.75
2 × 3	4.64	1½ × 2	2.40
3 × 3	5.12	2 × 2	3.52
3 × 4	6.08	2 × 3	4.64
3 × 5	7.04	3 × 3	5.12
4 × 6	8.00	3 × 4	6.08
6 × 8	9.12	3 × 5	7.04
		4 × 6	8.00
		6 × 8	9.12

[a] These are domestic sizes in inches.

Size

As noted under the price schedules, the prices are for domestic (U.S.) grading (sizes). Fig 2 is a chart of such sizes. To determine the grade of a pattern or block of mica, which has been trimmed ready for grading, place it over a chart drawn to the scale given in Fig 2 so as to include in the usable area the lower right corner of the 1 by 1 in. pattern. Then, determine the largest pattern completely included in the usable area of the block. The largest size shown in Fig 2 is 6 by 8 in. but some increase in price is generally provided for larger sizes and no mica should be trimmed smaller than is necessary to remove imperfections.

Fig 3 is a chart showing the corresponding grading to Indian and ASTM standard sizes. It provides some leeway in the length and width of each grade (size). A grade is determined on this chart by placing the block of mica to include the intersection of the bottom and right side lines of a chart drawn to scale as shown in Fig 3, as in Fig 2. Then select the grade (size) which completely covers a rectangle formed by the bottom and right side lines and the other two sides parallel to these and intersecting on or beyond the curve which forms the fifth side of the numbered area. For instance a block of mica would be grade 4 if it had a usable rectangle 2 by 3 in. because the far corner of such a rectangle reaches the curve of No. 4 grade. The minimum width of each grade is represented by the length of the left hand side of the grade area.

Punch mica must have a clear usable area of not less than 1 in. diam and the total area of the piece shall not be greater than five times the usable area.

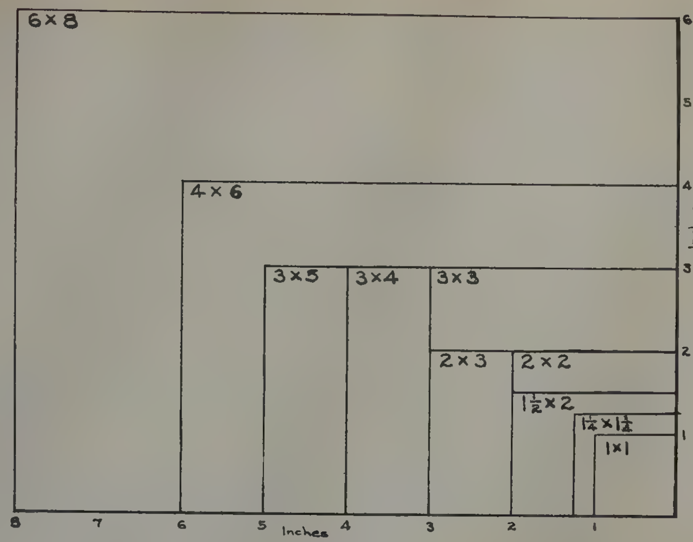


FIG 2—Chart for grading mica to domestic standard sizes.

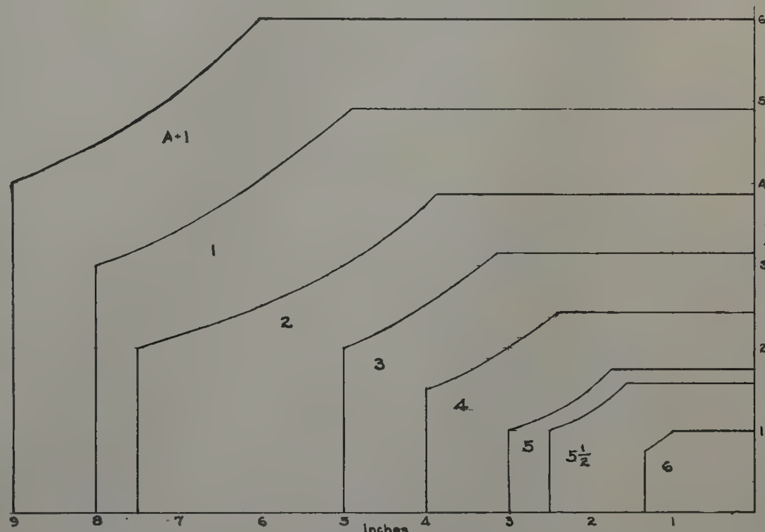
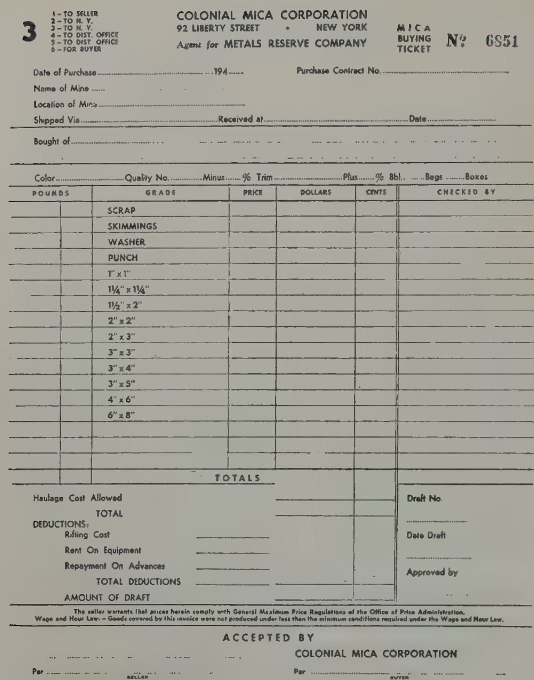


FIG 3—Chart for grading mica to India and ASTM sizes.

Domestic Quality

No. 1. Clear flat mica suitable for condensers.

No. 2. Light clay or vegetable-



3 1—TO SELLER 2—TO N. Y. 3—TO N. Y. 4—TO DIST. OFFICE 5—TO DIST. OFFICE 6—FOR BUYER

COLONIAL MICA CORPORATION
92 LIBERTY STREET • NEW YORK
Agent for METALS RESERVE COMPANY

MICA BUYING TICKET Nº 6851

Date of Purchase .... 194 ...... Purchase Contract No. ....
Name of Mine ......
Location of Mica ....
Shipped Via .... Received at .... Date ....
Bought of ....

Color .... Quality No. .... Minus ...... % Trim .... Plus ...... % Bbl. ...... Bags ...... Boxes

POUNDS		GRADE	PRICE	DOLLARS	CENTS	CHECKED BY
		SCRAP				
		SKIMMINGS				
		WASHER				
		PUNCH				
		1" x 1"				
		1¼" x 1¼"				
		1½" x 2"				
		2" x 2"				
		2" x 3"				
		3" x 3"				
		3" x 4"				
		3" x 5"				
		4" x 6"				
		6" x 8"				
		TOTALS				

Haulage Cost Allowed
TOTAL
DEDUCTIONS:
Rifting Cost
Rent On Equipment
Repayment On Advances
TOTAL DEDUCTIONS
AMOUNT OF DRAFT

Draft No.
Date Draft
Approved by

The seller warrants that prices herein comply with General Maximum Price Regulations of the Office of Price Administration.
Wage and Hour Law.—Goods covered by this invoice were not produced under less than the minimum conditions required under the Wage and Hour Law.

ACCEPTED BY
COLONIAL MICA CORPORATION
Per .... SELLER Per .... BUYER

FIG 4—Sample buying ticket.

stained, or light-spotted mica. This quality includes mica slightly curved or wavy.

No. 3. All mica below No. 2 quality commercially usable in sheet form. For description of Indian and ASTM Quality see references 3, 4, and 7.

Mica Buyers Report

The Mica Buyers Report (Fig 1) is used only for strategic mica. This report or "check sheet" is used by the buyer as a basis of arriving at the discount or bonus to be applied to the base price for any lot of mica. It contains a list of ten items to be considered and checked for every lot of mica on which a separate price is to be made. These ten items are:

1. *Color.* Mica is found in various shades and colors, the principal ones being shown in the left hand column of Table 2 and our grouping of them for check purposes as shown in the right hand column. The color of mica is more valuable as a means of identification than as an indication of quality.

Table 2 . . . Colors of Mica

Color	Group
White	White
Water color	White
Yellow	White
Amber	Ruby
Ruby or rum	Ruby
Red	Ruby
Brown	Ruby
Green	Green

We give no credit or make no discount for color. However, a sample of any very dark colored mica should be sent in to Asheville to be approved as strategic before any purchase is made of such mica.

2. *Thickness.* Check 100 pct if practically all the mica is 0.007 in. thick or thicker. Each buyer should carry a small piece of mica with him of the minimum thickness, so he can show the miner his basis of comparison. If approximately 5 pct of the mica is thinner than 0.007, check minus 5, if 10 pct, check minus 10. If more than 10 pct of the mica is less than 0.007, require the miner to reclass it and throw out the thins to be purchased separately as "thins" and price according to quality and preparation—½, ¾, or full trim.

3(a). *Spots* not (black). None, check 100 pct. Consider you will check minus 10 for the most spots the mica could contain and still be No. 1 quality, if it contained no other imperfection. Check minus 5 for a few spots but still enough to check the mica as light spotted. (See samples.) If too heavy spotted for minus 10 pct show under 10. Other defects.

3(b) *Clay stains.* None, check 100 pct. Very light, hardly noticeable, check minus 5 pct. Maximum clay staining permissable for No. 1 quality if it contained no other imperfections, minus 10 pct. (See samples.) If too heavy stained for minus 10 pct show under 10. Other defects.

4. *Air pockets.* None, check 100 pct. If 5 pct of area of the mica is air pockets, minus 5. If 10 pct of area of the mica is air pockets, minus 10. If more than 10 pct of the area of the mica is air pockets, send sample to Asheville for approval before buying. (Air creeps which work in from the edges during the trimming process are usually eliminated when the sheets are split for actual use, and should not be considered as a defect. Air pockets are the natural air stains which are in the mica when it is removed from the earth.)

5. *Trimming* (preparation). ½ trim is 100 pct, the basis of our prices. Trimming is defined as follows:

½ trimmed: minimum 2 adjacent sides full trimmed, no other trimming necessary, may contain some cracks but not in the pattern area. Maximum 60 pct outside of the pattern area. (Designated as North Carolina preparation).

¾ trimmed: trimmed practically all around, may contain few cracks on trimmed (not adjacent) sides but not in pattern area. Maximum 50 pct outside of the pattern. Must be trimmed all around and full trimmed on three adjacent sides.

Full trimmed: trimmed all around and all cracks and other defects trimmed out. Maximum 40 pct outside of the pattern.

¾ trimmed is plus 20 pct.

Full trimmed is plus 40 pct.

The buyer is to use his judgment as to the bonus for trimming and can give from 5 to 20 pct for ¾ trimming and 25 to 40 pct for full trimming, according to the actual quality of the preparation.

6. *Flatness.* Exact flatness is unusual in mica. It may be determined by noting the reflection of a straight line on the sheet of mica. Check 100 pct if you cannot readily see any curve or waviness. Check minus 5 pct if it is slightly curved or rippled. Check minus 10 pct if the curvature or rippling is more noticeable. Badly warped mica is nonstrategic and in cases where you have a question as to this qualification, send sample to Asheville. Waviness allowed for any quality is the hardest defect to determine, and maintain, on a uniform basis. Very few can agree on the actual amount of wave that can be tolerated for any use.

7. *Splitting.* Most mica is good splitting and can be checked 100 pct. If there is any tendency toward gumminess or cracking when the mica is split, check minus 5 or minus 10 according to the extent of this defect.

8. *Hardness.* Most mica is satisfactory as to hardness and can be checked 100 pct. If there is a tendency toward softness making the mica more fragile in thin sheets, check minus 5 or minus 10, according to the extent of this.

9. *Grading.* Check medium grading (100 pct) if the mica is of ample size to make the pattern. If as much as 5 or 10 pct is too small, check accordingly minus 5, minus 10 and write in remarks column the proper discount. Similarly, if 5 or 10 pct is oversize, check plus 5, plus 10 and write the proper bonus in remarks column. If more than 10 pct of the mica is undersize, require it to be reclassed before buying it. If more than 10 pct is oversize, permit the miner to reclass it before buying.

The quality of any piece of mica is determined by the largest pattern which can be cut from it. The total area of the piece should not be more than 2½ times the pattern.

10. *Other defects.* Write in here what they are and the discount you have applied for each such other defect. It has been agreed that the minimum discount for light black-spotted mica is 25 pct. (See samples)

DEFECTS NOT PERMITTED IN PATTERN

Cracks: ½ or ¾ trimmed mica may contain cracks but not in pattern area.

Hairlines: a hairline is a crack through some but not all the layers of a piece of mica. Hairlines are not permitted in full trimmed mica or in the pattern area in ½ or ¾ trimmed mica.

Sandholes: a sandhole is a hole in the mica usually filled by a grain of some other mineral like quartz or garnet. It is not permitted in full trimmed mica or in the pattern area of ½ or ¾ trimmed mica.

When a buyer is offered mica containing such defects in the pattern, he should class it down under these defects or refuse to buy it until these defects are trimmed out or classed out of it. Sandholes and hairlines are often very difficult to see and must be watched for carefully.

NO. 3 QUALITY NONSTRATEGIC MICA

Black-spotted or black-stained mica is only purchased by Colonial buyers when a miner has opened up a deposit with a view to obtaining strategic mica and it turns out to be nonstrategic, or, when a miner producing strategic mica encounters nonstrategic mica. The buyer will use his judgment and inform the miner a reasonable time in advance as to when he must discontinue the purchase of such nonstrategic mica, should it become too great a proportion of the miner's total mica production.

The OPA ceiling has been removed from nonstrategic mica, but our maximum prices for black-spotted and black-stained mica are the same as we were paying before the ceiling was removed as shown in Table 3.

Table 3 . . . Maximum Prices for Black-spotted and Black-stained Mica

Black-spotted		Black-stained		
Avg. Price	Size	Avg. Price	High	Low
0.08	Punch	0.06	0.06	0.06
0.40	1½ × 2		0.33	
0.65	2 × 2	0.45	0.52	0.40
1.00	2 × 3	0.65	0.82	0.60
1.30	3 × 3	0.80	1.05	0.80
1.60	3 × 4	1.00	1.30	1.00
1.85	3 × 5	1.30	1.50	1.30
2.35	4 × 6	1.60	1.90	1.40
3.00	6 × 8	2.00	2.40	1.75

General Instructions

Prices: discounts from the base price have been set up primarily to represent difference in quality and not a penalty to the miner. However, if a miner persists in overgrading, excessive thinness, poor trimming or any other practice that makes the mica difficult to buy, the buyer is warranted in making the discount sufficient so that the miner will find it to his advantage to improve his preparation of the mica.

Before you start on a buying trip, see that you have the necessary: scales, boxes, barrels, bags, tags, patterns for checking size, mica buyers reports, buying tickets, receipt books, drafts, pencils, and list of deductions.

First, examine at least 5 pct of the punch and each size of sheet mica.

Second, list on the back of the Mica Buyers Report the pounds of punch and each size of sheet you examined.

Third, make out the Mica Buyers Report (Fig 1.)

Fourth, make out the Buying Ticket (Fig 4), filling in all the top of the ticket and the prices according to the Mica Buyers Report. Then weigh the punch and put the weight in on the ticket. Weigh each size sheet and put the weight in on the ticket. Then extend the prices in accordance with the weights, total the weights and price extensions and note any deductions. (See sample ticket, Fig 4.)

Fifth, if there is a deduction, make out the deduction receipt, then the draft.

Sixth, pack up the mica for shipment.

Seventh, arrange for the punch and pattern to be shipped to the proper Colonial shop and show this on the ticket.

Your responsibility for a purchase is not complete until the mica has been delivered to the proper Colonial shop and all the reports, tickets, receipts, etc., have been completely made out and properly disposed of.

Your daily report should list purchases as follows:

From Ticket No. Draft No. Amount

These should be made out each night and mailed with the tickets the next morning.

The prices given in this guide were the actual prices paid by Colonial Mica Corp. at the time but are used only to illustrate the comparative value of different size and quality.

The Mica Buyers Report or check sheet could be used equally well with any other set of prices. However, should there be a radical change in prices the percentage of the discounts and credits might need to be changed.

Quality, price bonuses, and discounts are a desirable aid to maintain uniform quality and preparation. The price system recognizing differences in size and quality has been built up through years of experience and the publication of this Guide is only with a view to standardizing the procedure.

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Economics of Mineral Pigments

By W. M. MYERS,* Member AIME

Certain minerals possess inherent color and other properties that make them suitable for the pigmentation of paints, mortar, plaster, concrete, face brick, and other materials. Their production is one of the most ancient of mineral industries and their use by primitive man is well known. These natural pigments have some economic factors in their favor, particularly wide geographic distribution, low cost of production, ease of preparation, chemical stability, and permanence in use so that they continue to form the basis of a small but useful industry. When used in paints such pigments act as a filler and supply body and opacity in addition to color. Natural mineral pigments are prepared for the market by the comparatively simple operations of washing, grinding, blending, and calcining. Any one, or all of these processes may be applied to a single pigment. The development of synthetic pigments and the use of byproduct materials have introduced competition with the products of nature and are having a growing influence on the economic position of the mined minerals.

Development of the Industry

The industry developed to commercial stature in Pennsylvania largely because of the presence of beds of ocher extending from Easton in a southwesterly direction toward Reading, and a variety of small iron mines in the region. The mining of paint ores in the Lehigh Valley was active as early as 1850 and shortly after the Civil War the establishment of mills expanded production and led to the development of new products and eventually the manufacture of synthetic pigments. Pennsylvania continues to maintain its leadership as a producer of mineral colors. Production is recorded also in New Jersey, Illinois, Virginia, Ohio, and Georgia.

Iron as a Pigment

Iron in various compounds is the most important pigment in nature. The iron pigments are characterized by stability, light fastness, good covering power, high index of refraction, and economy in production. Ferrous compounds display a typical green coloration. This has not been employed as a commercial pigment to any great extent except incidentally in such greenstones as have been used in construction, or crushed and sized for application as a granule to roofing. Ferric oxides are characteristically red and form an important group of industrial pigments. Hydrated ferric oxides exhibit various shades of yellow and are present in such familiar pigments as ocher and sienna. The addition of small amounts of manganese oxide or carbon as organic matter to the yellows produces the brown umbers. Magnetite is one of the few black minerals that grinds to a black powder. The mineral pigments vary from earth colors of low intensity and great dilution of iron content to almost pure iron compounds with high tinting strength and correspondingly greater value.

There has been some relationship between the mining of iron ores and the production of mineral pigments; although the production of colors is not carried on appreciably as a byproduct of iron mining. A number of mineral pigment producers, particularly in the eastern states, have operated mines which in the past were small producers of hematite, bog ore, or carbonate ore. The knowledge of the location of the deposit and its possibilities as a producer of pigment are due to its past history as a producer of metallic ore.

Many iron compounds may be changed in color by calcination, the end product being Fe_2O_3, a stable red oxide. The yellow hydrated oxides may be changed to red. The gray carbonate, found in Carbon County, Pa., calcines to a fine red product. Ferrous sulphate (copperas) often available in tonnage as a byproduct of the steel industry, breaks down under heat or chemical action leaving a residual red oxide. Pigments produced by calcination or dissassociation of compounds possess the advantage of fine particle size and ease of grinding.

Mineral Pigments

MINERAL BLACKS

Many black minerals grind to a light colored powder. This characteristic is familiar to the mineralogist as the white streak produced by a black

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mineral. A number of rocks including black slate and shale, high in carbonaceous matter, have been ground to produce pigments low in coloring strength. The semianthracites of Sullivan County, Pa., grind to a black useful for certain purposes. The market for blacks is dominated by the more powerful lamp and carbon blacks. Precipitated magnetic blacks supply synthetic competition in some markets.

MINERAL BROWNS

The siennas and umbers and other natural brown oxides owe their characteristic color to the inclusion of darkening agents which alter the typical yellow or hydrous iron oxides. Calcination is frequently employed to produce "Burnt" sienna and umber. The calcination process brightens the color by removal of carbonaceous material and enhances the red tones by increasing the amount of ferric oxide. Pure browns of high iron content possess high tinting value and compete with the natural products.

MINERAL REDS

The production of mineral reds is the most important division of the mineral pigment business. As will be noted in Table 1, in 1947 the reds accounted for 57 pct of the total tonnage of the industry and 66 pct of the value. The most important item is the pure red oxides mostly produced by the calcination of very pure yellow hydrated oxides. These yellow oxides have been prepared from a solution of metallic iron; by precipitation from a solution of ferrous sulphate, or by treatment of other materials. Reds are also produced by calcination of an iron carbonate ore, mined in Carbon County, Pa., from a seam averaging about 1.5 ft in thickness. Selected hematites supply natural red oxides. Venetian red, produced by the calcination of ferrous sulphate and lime, consists of a mixture of finely divided iron oxide and calcium sulphate.

MINERAL YELLOWS

Ocher is essentially an iron-stained clay and its iron content varies through a wide range of color and tinting strength. Natural yellow oxides with a high iron content are prepared by grinding selected materials, generally iron ores of the limonitic type. Pure yellows are prepared by precipitation of hydrated oxide from iron bearing solutions prepared from metallic iron and ferrous sulphate.

Magnitude of the Industry

The magnitude of the industry is shown by Table 1.

Table 1 . . . Natural Mineral Pigments and Manufactured Iron Oxide Pigments Sold by Producers in the United States, 1947

Data From the U. S. Bureau of Mines

	Short Tons	Value
Mineral blacks	[a]	[a]
Precipitated magnetic blacks	[a]	[a]
Natural brown oxides	5,861	$308,440
Vandyke brown	[a]	[a]
Pure browns (96 pct or better iron oxides)	1,016	219,686
Natural red oxides	20,524	946,997
Pure red oxides (98 pct or better Fe_2O_3)	17,331	3,481,083
Venetian reds	7,127	579,603
Pyrite cinder	1,682	110,863
Other red iron oxides	18,817	2,214,358
Natural yellow oxides (high Fe_2O_3)	[a]	[a]
Pure yellows (85 pct or better Fe_2O_3)	10,496	1,635,365
Ochers (low Fe_2O_3)	9,130	213,133
Siennas:		
Burnt	940	141,943
Not burnt	1,441	201,493
Umbers:		
Burnt	3,051	322,688
Not burnt	671	61,443
Other	17,280	730,066
Total	115,367	11,167,161

[a] Included under "Other."

FOREIGN IMPORTS

Mineral pigments imported from foreign countries occupy a prefered position in some industries. This is because of careful preparation, inherent superior qualities of color, and tinting strength. Exact equivalents of imported materials have not been found in the United States in the natural state. Among these imports should be mentioned Persian reds, Spanish reds, French ochers, Italian sienna, and umber from Cyprus and Turkey. The quantities imported are not large and have been subject to interruptions due to wartime conditions.

RECOVERY OF HYDROUS IRON OXIDES FROM COAL MINE DRAINAGE

The presence of hydrous iron oxides in streams receiving mine waters in the bituminous and anthracite areas is a common occurrence. The beds of many streams in the bituminous district of western Pennsylvania are yellow for miles. The appearance of the streams and their utility for recreation is destroyed. The yellow material is locally known as "sulphur mud." The collection of this material in a few favorable localities has become the basis of a small mineral industry and has supplied a new material for mineral pigments.

This material originates from the oxidation of pyrite and marcasite in the coal. It does not take place until mine workings have exposed the coal permitting oxidation of these sulphides. The products of oxidation are ferrous sulphate and sulphuric acid. The ferrous sulphate being a soluble salt is carried off with the acid in the mine drainage. This solution is generally clear and precipitation does not occur commonly in quantity in the mine. It cannot take place until the ferrous sulphate solution has been oxidized and mine conditions are not favorable. Oxidation and precipitation begin promptly as the waters leave the mine. It is commonly noted that the mine drainage may be clear until merging with the oxygen-charged surface waters in adjacent streams after which precipitation is rapid. The chemistry is variable represented by the following pattern:

$$FeS_2 + 7O + H_2O = FeSO_4 + H_2SO_4$$
$$4FeSO_4 + 2O + 7\,H_2O = 2Fe_2O_3{\cdot}3H_2O + 4\,H_2SO_4$$

The $2Fe_2O_3{\cdot}3H_2O$ is precipitated as an amorphous substance. Petrographic examination and confirmation with the X-ray shows that the yellow hydrated oxides recovered for conversion to pigment are amorphous. Therefore, they are classified as limonite, a hydrous hydrated ferric oxide sometimes represented by the formula, $Fe_2O_3{\cdot}H_2O{\cdot}nH_2O$.

In a few favorable localities the topography is such as to supply a natural settling basin in which the mine waters have been confined long enough to permit the accumulation of thousands of tons of material. In some cases recovery of the limonite has been made from such areas after exhaustion of the coal had led to the closing of the mine.

In western Pennsylvania, where pigment recovery is attempted, logs and rocks are placed in the discharge of mine waters pumped from the mine. These obstacles agitate the water and aerate the solution accelerating precipitation of the iron oxide. To permit economical recovery there must be a concentration of a substantial tonnage within a few hundred feet, of sufficient thickness to permit recovery by ordinary shoveling. The percentage of the total iron content deposited is doubtless small and much of the material

escapes to be left in the streams in unrecoverable form. The possibility of making a complete recovery has been suggested to prevent stream pollution and to recover a valuable industrial material.

The bright yellow mud is shoveled from the floor of stream bed and basin and exposed to the sun to dry and then trucked to the processing plant or to cars for shipment by rail. As shipped, the material contains about 40 pct moisture. The material as precipitated is remarkable pure but contamination with soil, sand, and vegetation is difficult to prevent. Cars of material analyzing 96 pct hydrous iron oxide have been shipped; material under 80 pct is not acceptable. Lack of uniformity in shipments is objectionable. A small royality per ton is paid in some cases to the mine operators. The average price paid to the producer has been $6 a ton. No record of total recovery has been kept, but estimates indicate that over 10,000 tons have been recovered.

These hydrous iron oxides have not been used as yellow pigments to any extent. The bulk of the material has been calcined in furnaces maintaining an oxidizing atmosphere and then ground in a buhr mill, rolls, Raymond mills or other equipment. The final product is a beautiful red possessing the assets of high quality iron pigments.

ECONOMIC CONSIDERATIONS

The trend has been toward the manufacture of high grade synthetic compounds of great tinting strength to replace natural pigments. The synthetics now contribute nearly one half the tonnage and three quarters of the dollar value of the entire industry. The tinting strength of the synthetics may be ten times or more that of the natural pigment. Less weight is required, transportation charges are less, and the uniformity of the product permits accurate proportioning. As the synthetics are produced under accurate technical control they can compete with the highest quality natural pigments imported from foreign sources in the past. The importance of imported materials tends to decline. Natural pigments possess the advantage in some uses of supplying bulk as well as color. In instances where the final product is sold at a higher price than that paid for the pigment there may be a profit in the resale of the pigment. Therefore, the use of a natural pigment with its lower tinting strength may be advantageous as more pounds must be used to produce the desired color. Apparently the natural pigments have stabilized their position in certain industries where their use is feasible and will continue to hold a place in the industry. New markets are likely to be dominated by the more powerful synthetics.

Petrology of High Titanium Slags

Abstract

(The paper in its entirety is published in *Transactions AIME*, **185**, 914–919: *Journal of Metals*, December 1949. TP 2714 D.)

By CHARLES H. MOORE, JR.* Member AIME, and H. SIGURDSON*

When lime and magnesia are used as fluxes in the smelting of titaniferous ores fluid, digestible slags low in iron oxide and high in titanium dioxide are produced. The mineral phases present in such slags are $(Fe,Mg)O \cdot 2TiO_2$, $CaTiO_3$, and minor amounts of silicate glass (or its devitrification products). The flux additions are added to the ore in relative amounts to crystallize a ratio of $(Fe,Mg)O \cdot 2TiO_2$ to $CaTiO_3$ of near the eutectic proportions of 60 pct $(Fe,Mg)O \cdot 2TiO_2$ and 40 pct $CaTiO_3$.

$FeO \cdot TiO_2$ (ilmenite) remains present in the slag until the FeO content is reduced to approximately 8 pct. Optimum slag composition is obtained by reducing the FeO content to between 2 and 6 pct. Continued reduction shifts the $(Fe,Mg)O \cdot 2TiO_2$ to oxygen deficient structures classified as $MgTi_3O_6$ and $MgTi_2O_4$. Further reduction yields titanium oxycarbides and freezes the melts.

The $(Fe,Mg)O \cdot 2TiO_2$ structure shows a complete isomorphous replacement of iron by magnesium. It also forms a solid solution compound with as high as 12 mols of TiO_2 under reducing conditions without the occurrence of rutile or reduced titanium oxides in the slag.

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